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ADVERTISING RATES GIVEN ON APPLICATION.

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CONTENTS.

Editorial 211
International Electrical Congress..... 214
The Metallurgy of Gold on the Rand Gold Fields. By Clement Dixon and M. Torrente 215
Electrolytic Refining of Gold. By Emil Wohlwill..... 221
Analysis of Refined Copper. By L. Addicks..... 224
Studies on the Siemens Process for Winning Copper. By M. DeK. Thompson, Jr. 225
Wet Methods of Extracting Copper. By C. H. Jones..... 232
Palladium 234
Recent Metallurgical Patents 235
Analysis of Current Electrochemical Patents. By George P. Scholl. 236
Synopsis of Articles in Other Journals..... 243
Current Notes 249
Faraday Society Meeting 249
Correspondence: Electrochemical Notation 251
Book Reviews 251
Two Stories of Zinc..... 252
Electrolytic Interrupter 252
Sulphur Furnace 253
Personal 253
Industrial Notes 253
Digest of United States Patents Prior to July, 1902. By Byrnes & Townsend. (Production of Aluminium from Fused Electrolytes). 253

THE METALLURGY OF GOLD ON THE RAND.

In this issue we publish an article by Messrs. C. Dixon and M. Torrente which our readers will find to be one of the most valuable and important contributions to the literature on the metallurgy of gold on the Rand. It is a very concise and lucid exposition of the development of the cyanide process in that classical country of gold metallurgy, written by two men who have actively participated in its development. Mr. Dixon is a member of the Council of the Chemical, Metallurgical and Mining Society of South Africa, while Mr. Torrente has been most intimately connected with the development of the electrolytic precipitation process during recent years. This makes the statements of the authors authoritative. On the other hand, with respect to that special problem which is of particular interest to our readers—the question of electrolytic precipitation versus zinc precipitation—the exposition of the authors will be found most interesting and impartial. In the statement of facts the article corrects two misconceptions which have repeatedly found expression. First, we refer to a statement recently made by a strong and able advocate of zinc precipitation; this statement, made in one of our contemporaries, was to the effect that electrolytic precipitation is now out of the race in South Africa. Surely, a process by which, during the last year, 239,385 tons of sands or tailings, and 129,137 tons of slimes were treated, with a total production of 72,164 ounces of gold, cannot be considered as dead. Secondly, the Andreoli process, instead of having superseded the Siemens & Halske process, as has been stated repeatedly, has failed in practice to improve matters with the working solutions on the Rand.

The history of gold metallurgy is a good example of the general rule that chemical methods are only resorted to after purely mechanical methods of treatment and concentration have spoken the last word. As long as placer mining pays well, the miner will not care for the application of chemistry for winning gold. But in every branch of metallurgy there comes a time when the miner realizes that he must do something more than apply his old mechanical methods of concentration. He will then try chemical methods, and more than once in the history of metallurgy the whole aspect of an industry has been revolutionized by the introduction of a chemical method. This is what the cyanide process has done for the Rand gold fields, since it enabled many low-grade propositions that could otherwise never have been worked at all to pay handsome dividends. And again, the average chemical engineer of the past did not resort to electrochemical means, unless he was forced to do, in order to accomplish a certain special end. In gold metallurgy it was the necessity to precipitate the gold from very dilute cyanide solutions which led to electrolytic precipitation; and the

merits of the electrolytic process are not impaired by the fact that it later has become possible to use a modified form of zinc precipitation for the same purpose. While thus chemical methods come after purely mechanical methods and electrochemical methods after purely chemical methods in a metallurgical industry, it would be absolutely wrong to believe that the later methods supersede the older ones completely; that the chemist supersedes the mechanical engineer, and the electrochemist supersedes the chemist. This is not the case; what happens is that the chemist and electrochemist combine their knowledge with that of the mechanical engineer, to lead a process of metallurgical treatment to greater perfection. This is quite evident from the article of Messrs. Clement and Torrente. Much of the progress made in the cyanide process in recent years in South Africa has been due to the invention of labor-saving apparatus and, in general, to the fact that the mechanical devices used in the process were brought to a high state of perfection. As Mr. Herreshoff emphasized in a paper published in our last issue, the keynote of progress in chemical engineering is that a chemical engineer shall not be a chemist pure and simple, but shall have had a sound training in mechanical engineering. To insure that the desired chemical or electrochemical reaction takes place, he must have the materials to be treated in the right conditions—in the right shape, at the right temperature, etc.—and for working continuously and on a large scale, this requires a good, sound knowledge of mechanical engineering.

The development of the cyanide process shows that competition is a good thing, that it stimulates a healthy rivalry. It is certainly due to the rivalry between the advocates of zinc precipitation and electrolytic precipitation that both processes were continually modified and improved. The history of gold in South Africa also shows that from a business point of view —and this is the only correct one for judging an engineering enterprise—the better process is not necessarily the one which is technically more perfect. Other considerations may be of greater importance. Thus, in 1894, the advantages were in favor of the electrolytic process, for which royalty of 2 per cent was charged, against 10 per cent royalty for the McArthur-Forrest process. Then the McArthur-Forrest patents were upset, the 10 per cent royalty were no longer a factor in the case, and the advantages were in favor of zinc precipitation. It is only fair to say that at present zinc precipitation seems technically more perfect than electrolytic precipitation; the main disadvantage of the latter is that the solutions leaving the boxes always carry from 0.5 to 1 dwt. of gold per ton, and that it has so far been impossible to reduce this minimum. On the other hand, it is also fair to state that zinc precipitation in the improved form now used on the Rand—*i. e.*, the zinc-lead couple—is distinctly an electrochemical phenomenon; the essential difference in the physics of the processes is that with electrolytic precipitation the electric current is sent through the boxes from an external source, while with the zinc-lead couple we have in the box itself very numerous little galvanic couples or short-circuited galvanic cells, which generate the current, with zinc as anode and lead as cathode. The decisive point which marked the most important improvement of zinc precipitation—Betty's method of adding free cyanide solution at the head of the precipitation boxes—is also quite analogous to a trick often applied in electrolysis by electrolytic refiners or electroplaters who, in order to keep their anodes bright, add free acid to the bath; Dr. Wohlwill's article on gold refining, published in this issue, may be cited as an example.

EFFICIENCY

To operate a process with a high efficiency is the keynote of every engineering success. But what is meant by efficiency? This term is used in different cases in a different sense, and it would be well if an author would always state exactly what kind of efficiency he means. In any case, if one speaks of efficiency, one compares the quantity of something that is put into the operation of a process, with the quantity which one gets out of it. In electrolytic processes we may compare the quantity of electric current—measured in coulombs or ampere-seconds; or, what seems more feasible in engineering work, measured in ampere-hours—with the quantity of the material which is reduced at the cathode or oxidized at the anode. Between both quantities there is a strict proportionality according to Faraday's law, which may be considered as exactly correct, and the factors by which one quantity, *e. g.*, the number of ampere-hours, must be multiplied in order to get the other quantity, *i. e.*, the grams, are well known. For example, we know that in electrolysis of a monovalent copper salt one ampere-hour deposits 2.37 grams of metallic copper, according to Faraday's law. We call this the theoretical figure, because we make the silent supposition that the only reaction which occurs at the cathode during electrolysis is the reduction of monovalent copper ions to metallic copper. In practice, however, it is not often that we have to do simply with one reaction. What takes place in practice is generally a complex reaction. For instance, in our example of copper deposition, we may have, simultaneously with the reduction of the cuprous ions, an evolution of hydrogen. Since we want to get metallic copper, we consider the ampere-hours spent for the evolution of hydrogen as wasted. We compare the amount of copper which we really get with the amount of copper which we should get according to Faraday's law, and by expressing the former in per cent of the latter, we speak of the ampere-hour efficiency of an electrolytic process. This ampere-hour efficiency is necessarily always below 100 per cent, since 100 per cent corresponds to the ideal condition that only the one desired reaction really occurs.

The calculation of the ampere-hour efficiency is of great value for the thorough understanding of the working of an electrolytic process, because this figure tells us to which degree the one reaction which we desire to obtain really goes on. But the ampere-hour efficiency is not of direct practical value, since what the operator of an electrolytic process has to pay for is not the ampere-hours consumed, but the watt-hours consumed. He is therefore more interested to know to which degree the electrical energy which is consumed is utilized for the production of the chemical effect he desires to obtain. For this purpose we have the watt-hour efficiency, which is the quantity of material which we really get, in per cent of the theoretical quantity which we should get, if the electrical energy was completely changed into chemical energy, and used for nothing but the one effect which we want to obtain. While the am-

pere-hour efficiency is essentially applicable to electrolytic processes only, the term watt-hour efficiency can be applied to any electrical process, and therefore comprises in electrochemistry not only electrolytic, but also furnace processes. In the ratio representing the watt-hour efficiency there is again a "theoretical" figure. This theoretical figure is generally based on the assumption that in the process under discussion we have to do with two kinds of energy only, viz., electrical energy, which is consumed, and chemical energy, which is produced. This method of calculating the watt-hour efficiency is open to theoretical objections for the reason that in practice heat energy enters into every electrochemical process. For this reason, the watt-hour efficiency, although nearly always below 100 per cent, may become higher than 100 per cent. This will be the case in such processes in which not only electrical energy is consumed, but also heat obsorbed from the surroundings, to produce the desired chemical effect. From a theoretical point of view, it would be preferable if we would not base our calculation on the thermochemical value of the heat of reaction, but on Helmholtz's "free energy" of the reaction. For a longer discussion of this point, we may refer to an editorial in our January issue.

As long as we consider an electrochemical process by itself, the figures for ampere-hour efficiency and watt-hour efficiency give us important information on the working of the process. In an electrochemical laboratory it will always be the highest aim of the experimenter to get as high an ampere-hour efficiency as possible. In practice, however, an electrochemical process is only one step in the whole manufacturing operation, and what is most important here is to get as high a financial efficiency as possible. Financial efficiency is the selling price of the products in per cent of the money to be put into the process. In practice on a large scale, it may not pay, under certain circumstances, to get a very high ampere-hour efficiency or watt-hour efficiency, if this can be obtained only by special attendance or handling of the materials. A reaction with a very low watt-hour efficiency may be a practical or financial success; lighting with electric incandescent lamps is an example. As long as the cost of power is the most important item on the expense sheet of a process, a high watt-hour efficiency is evidently of the greatest importance; but it is equally clear that it should not be obtained by means which introduce new items of expense in the cost sheet, if they are greater than the saving in power.

PLATINUM RECOVERED FROM GOLD.

Electrochemical methods in their application to metallurgy have proven most successful in the latest stage of metallurgical treatment, when the problem is to get a very pure—and for this reason a relatively high-priced—product from materials which have already undergone previous metallurgical treatment and purification. In this way the electrolytic refining of copper and nickel has been specially successful. Electrolytic copper refining methods not only yield very pure copper, but they also enable us to recover the precious metals contained in the copper. The copper refineries of this country produced in 1902, besides 279,000 tons of electrolytic copper, 27,000,000 ounces of silver and 346,020 ounces of gold. For parting the gold from the silver, electrolytic methods have also proven successful, and the same is the case with the final problem to refine the gold and recover its valuable impurities, like platinum.

The Wohlwill electrolytic process of gold refining is in successful use in the Philadelphia Mint, and our January issue of 1903 contained a very interesting description by Dr. Tuttle of the process, as operated in Philadelphia. We are pleased to publish in this issue the first of two articles by the inventor, Dr. Wohlwill, in which the author emphasizes the great simplicity of recovering platinum, iridium, etc., which apparently always occur in Nature together with gold and silver. Dr. Wohlwill's notes deserve the most careful consideration of the metallurgists of this country.

COPPER PRODUCTION FROM THE ORES

In this issue we publish two interesting articles on the production of copper from the ores. In one of them, Mr. Charles H. Jones describes at some detail the methods which are in use at Rio Tinto in Spain for leaching and extracting copper. While the formulæ by which the author of the paper represents the reactions may be open to discussion, his very full description of the actual work of the process on a large scale should be of great interest, since much of the information published before on this subject has become obsolete in the meanwhile. The process is in no way electrochemical in the sense that electrical energy is expended from an external source; on the other hand, a complete understanding of the reactions can only be obtained from the point of view of an electrochemist who sees electrochemical phenomena going on without the expense of external electrical energy, whenever there are metallic materials in contact with liquids representing short-circuited galvanic cells.

A very interesting paper by Dr. M. DeK. Thompson, Jr., also published in this issue, refers to the Siemens electrolytic process for winning copper from the ores. This process is of great historical interest, since it was probably one of the very first ever proposed in metallurgy for winning metal from the ores by electrolytic means. As far as we know, the process has never been used on a large scale, nor has any exact information been published about the operation of the process. Mr. Thompson's paper is most valuable, since it gives the results of a very careful and extended laboratory research on the working of the process. In this respect the information given in the paper appears to be complete, and should be of value to copper metallurgists, although they may not be interested in the Siemens process. The paper does not enable one to draw any conclusion concerning the reasons why the Siemens process has not been a commercial success, but this does not diminish the value of the paper. A laboratory investigation is different from an operation on a commercial scale. In commercial practice in the laboratory one works on a test-tube scale, and it is easy to provide the proper conditions for the desired electrochemical reaction. On the other hand, in metallurgical practice on a large scale, the problems of handling the materials, of putting them into the correct condition for starting the reaction and—last, not least—maintaining the materials in proper condition during continuous operation, are often most difficult, and, what counts more in practice, are often very expensive.

INTERNATIONAL ELECTRICAL CONGRESS.

Preliminary programmes have just been issued by the secretaries of Sections A, B and C of the International Electrical Congress to be held in September in St. Louis.

Section A.

Dr. E. L. Nichols is the chairman and Dr. H. T. Barnes the secretary of this section, which deals with "Theory." The following papers have been promised by foreigners:

A Paul Drude, "Metallic Conduction."
W. Jaeger, "Electrical Standards,"
Oliver Lodge, "Ions."
H. Nagaoka, "Magneto-Striction."
J. J. Thomson, subject to be announced.
J. S. Townsend, "The Theory of Ionization by Coliision."
J. Violle, "Secondary Standards of Light."
C. T. R. Wilson, "Condensation Nuclei."
P. Zeemann, "Magneto-Optics."

The following papers have been promised by American scientists:

H. T. Barnes, "The Mechanical Equivalent of Heat as Measured by Electrical Means."
Carl Barus "Atmospheric Nuclei."
Louis A. Bauer, "The State of our Knowledge Regarding the Earth's Magnetism."
D. E. Brace, "Magneto-Optics."
H. S. Carhart and G. W. Patterson, Jr., "The Absolute Value of the Electromotive Force of the Clark and Weston Cells."
C. D. Child, "The Electric Arc."
K. E. Guthe, "Coherer Action."
E. P. Lewis, "Electrical Discharges in Gases."
L. T. More, "Electrostriction."
E. Fox Nichols, "The Unobtained Wave-Lengths between the Longest Thermal and the Shortest Electric Waves yet Measured."
E. L. Nichols, "Standards of Light."
Harold Pender, "Magnetic Effect of Moving Charges."
M. I. Pupin, "Electrical Theory."
Edward B. Rosa, "Alternating Current Measurements."
E. Rutherford, "Radioactive Change."
J. C. McLennen, "Radioactivity of the Atmosphere."
J. Trowbridge, "Electrical Discharge in Gases."
A. G. Webster, "Electrical Theory."

Section B.

This section deals with "General Applications," the chairman being Dr. C. P. Steinmetz, and the secretary, Dr. Samuel Sheldon. The following papers have been announced:

E. Arnold, "Direct-Current Commutation."
O. S. Bragstad, "Compensated Alternators."
R. E. Crompton, "Standardization of Dynamo-Electric Machinery and Apparatus."
Andre Blondel, "Calcul des Alternateurs."
J. Elster and O. Geitel, "Uber die Natürliche Radioactivität der Atmosphäre und des Erdbodens."
C. Feldmann, "The Distribution of Voltage and Current in Closed Conducting Networks."
A. Heyland, "Self-Regulating and Compounded Synchronous Machines."
W. M. Mordey, subject to be announced.
A. Nodon, "Rectifiers."
W. Preece, "Electricity in Ancient Egypt."
C. A. Adams, "Magnetic Leakage in Alternating-Current Machinery."
C. Day, "Electric Motors in Shop Service."
J. W. Esterline, "Carrying Capacity of Cables and Conductors."
H. W. Fisher, "Sparking Distances, Corresponding to Different Voltages."
H. J. Ryan, "The Design of Insulators."
D. B. Rushmore, "The Regulation of Alternators."
E. B. Rosa, "The Influence of Wave Shape upon Alternating-Current Meter Indications."
Clayton H. Sharp, "The Equipment of a Commercial Testing Laboratory."
H. B. Smith, "Very High Voltage Transformers."

Section C.

This is the electrochemical section of the Congress, the chairman being Dr. H. S. Carhart and the secretary Mr. Carl Hering. The following papers have been announced:

S. Arrhenius, "Methods of Determining the Degree of Dissociation."
W. Borchers, "Electrometallurgy of Nickel."
Sherard O. Cowper-Coles, "Electrolytic Methods for the Rapid Production of Copper Sheets and Tubes."
F. Dolezalek, subject to be announced.
J. Sigfried Edstrom, "Electrical Extraction of Nitrogen from the Air."
H. Goldschmidt, "Aluminothermics."
F. Haber, "Electrolytische Zerstörungen in der Erde."
P. C. L. Heroult, "Electrometallurgy of Iron and Steel."
J. Swinburne, "Chlorine Smelting."
W. D. Bancroft, "The Chemistry of Electroplating."
A. G. Betts and Edward Kern, "The Lead Voltameter."
H. S. Carhart and C. A. Hulett, "The Preparation of Materials for Standard Cells."
Thomas A. Edison, "Alkaline Batteries."
K. E. Guthe, "The Silver Voltameter."
Carl Hering, "The Units Employed in Electrochemistry."
J. T. Morrow, "Notes on Electrolytic Copper Refining."
J. W. Richards, "The Energy Absorbed in Electrolysis."
T. W. Richards, "The Relation of the Theory of Compressible Atoms to Electrochemistry."

The papers given above have already been promised. The following papers are also expected in the class of electrochemistry, but no definite promise has yet been received from the authors:

M. Berthelot, "Thermochemistry and Electrochemistry."
W. Hittorf, "Transport Numbers of Migration."
R. Lorenz, "Electrolysis of Fused Salts."
H. Moissan, "The Electric Furnace."
W. Ostwald, "Electrolytic Theory."
W. Nernst, "Theory of Solutions and the Dissociation Theory."
H. W. B. Roozeboom, "Phase Rule in Electrochemistry."
W. D. Bancroft, "The Chemistry of Electroplating."
D. H. Browne, "Electrolytic Production of Copper and Nickel."
B. Magnus, "Electrolytic Copper Refining."
T. L. Wilson, "Calcium Carbide."

Electrometallurgy of Iron and Steel.—A paper on this interesting and timely subject is to be presented by Messrs. F. A. J. FitzGerald and P. McN. Bennie before the "American Foundrymen's Association" in Philadelphia, on June 1.

The Calculation of Line Batteries.—In a paper recently presented by Mr. W. E. Winship before the American Institute of Electrical Engineers, the author developed formulas for the calculation of storage batteries to be used in tramway work. In this case the storage battery is discharged to supply the energy for the peaks of the load curve, while it is charged when the load is below the average; at average load it is a "floating" battery. The capacity should be reckoned approximately at the rate corresponding to the duration of all the net discharges and not at the rates at which the separate discharges occur. Comparative calculations should be made, considering, first, the cost of copper alone, and secondly, the cost of copper and batteries, for equivalent line voltages, also the cost and depreciation in the two cases together with the transmission losses. Credit is to be given to the battery for the decrease of cost of power, due to load equalization, and for its possibility of being used as a reserve in case of breakdown.

THE METALLURGY OF GOLD ON THE RAND GOLD-FIELDS.

BY CLEMENT DIXON AND M. TORRENTE.

So much has been written on this subject already that the paper now presented to ELECTROCHEMICAL INDUSTRY may savor somewhat of "ancient history" and repetition. At the same time the authors venture to say that there are many interested readers of this journal who will welcome a resumé of the metallurgical practices on these fields. We will first briefly refer to the work done here in the "early days" and follow the rapid advancement made up to the present time, when it will be seen what splendid progress has been made since the first mill—consisting of only five stamps—was erected on these fields in 1887, only seventeen years ago. From this date the flotation of companies and erection of mills proceeded with great energy.

Up to the year 1890 the gold recovery methods consisted simply of wet crushing, with 750-pound stamps and ordinary plate amalgamation followed by various concentration devices, such as buddles, blanket strakes, shaking tables, canvas tables, vanners, etc.

The oxidized banket ore, however, was not easily amenable to concentration and the percentage of concentrates caught was small. These concentrates were worked in various kinds of amalgamation pans or stored for future treatment. The total gold recovered in those days probably did not amount to more than 60 per cent of the original value of an ore, then going about an ounce to the ton or over.

The year 1890 marked a great advancement in the metallurgy of gold on the Rand, for both the chlorination process and MacArthur-Forrest cyanide process were introduced. The first chlorination plant was a Newbery-Vantin, but it was not very successful, and soon suspended operations. In the following year, 1891, the first Plattner chlorination works on these fields was erected on the Simmer and Jack mine for the treatment of concentrates, and plants of a similar type were afterwards erected and worked with much success on the Robinson Gold Mine and at the Rand Central Ore Reduction Co.'s works. The cyanide process, first introduced experimentally in May, 1890, however, was an indisputable success from the start, and at the end of the year the first plant—to work tailings by this process on what was then considered a large scale—was erected on the Robinson Gold Mine.

The vats were square wooden ones of about 40 tons capacity. The working cost on this plant in 1891—treating accumulated tailings—were fully 12/ per long ton and the consumption of cyanide was about 3.32 pounds per ton of tailings treated. The strength of solution used was about 0.5 per cent of KCy, and the actual extraction about 65 per cent of the original value of tailings averaging nearly 14 dwts. Caustic potash was the agent used to neutralize the acidity of the weathered tailings.

The introduction of the cyanide process revolutionized the whole aspect of the industry here, and enabled many low-grade propositions, that could otherwise never have been worked at all, to pay handsome dividends. The process has resulted in the mill now being regarded mainly in the light of a pulverizer, rather than a gold extractor, the percentage of gold caught by amalgamation (which is approximately 60 per cent from say 11 dwt. ore) not being the most important factor in the operations of to-day. The mechanical devices for the handling of tailings have been rapidly improved, and the old method of discharging the residue tailings by laborously shoveling them over the sides of the vats into trucks soon gave way to bottom and side discharge doors, and the capacity of the vats have been increased from 40 tons to 400 tons—the original square wooden vats being supplanted by huge circular steel ones.

To-day, instead of the sands and slimes being first run into dams and there filled into trucks and hauled back to the cyanide treatment vats, the method now in vogue is that of "*direct filling*", *i. e.*, the pulp as it comes from the mill runs into a sump from which it is at once elevated by a huge tailings wheel or pumps to the desired height above the cyanide vats to allow of all future operations being conducted by gravity. From the elevator the pulp is conveyed along a launder where it is automatically sampled and passes into hydraulic classifiers called Spitzlutten. These are boxes so constructed that the heavier sands and concentrates (amounting to about 10 per cent of the total pulp and assaying about 12 dwts. to the ton from an original ore averaging say 11 dwts. per ton) are separated from the remainder of the pulp, which are led off direct into vats for special treatment (either by cyanide or chlorination) while the lighter sands and slimes continue their course down the launder—where they are again automatically sampled—and then pass through a spitzkasten, which divides the pulp again into two products—"leachable sands" and "slimes."

A spitzkasten is an hydraulic classifier so constructed that by a definite pressure of water being introduced (in some cases, however, no extraneous water is required) at the bottom of the box, the slimes are borne upwards and over the edge of the box and flow through a further series of spitzkasten to the slimes vats, while the sands are drawn off through a pipe at the bottom, and are led away to the percolating vats. The sands product would represent approximately say 65 per cent of the elevated pulp and assay about 4 dwts.—the remaining 25 per cent of the plup being slimes assaying, say, 2½ dwts.

SANDS

The sands from the spitzkasten are conducted direct to collecting vats—super-imposed over the leaching or treatment vats, proper—and there deposited either by means of Butters & Meins rotary distributors, or by a large flexible hose which is manipulated by a Kaffir stationed in the vat who soon becomes experienced in obtaining the desirable even distribution of the sands throughout the vat. In the process of filling by either means the surplus water is continually escaping from the collecting vat, in the one case into a peripherial overflow launder, and in the other case through a side discharge door, the height of which is regulated by slats of wood and canvas aprons as the level of the sand rises in the vats.

When the collecting vat is filled with sand and the water drained off, a preliminary weak cyanide solution (say 0.1 per cent to 0.15 KCy and in the proportion of, say, 1 part of solution to 5 of ore) may be pumped on and allowed to percolate through the mass and through the filter mat at the bottom. The first part of this leached solution is sent to the weak wash sump to be used for final washings at end of treatment of a charge, while the remainder of this solution, as soon as it comes through with sufficient gold to warrant it, is sent to the gold extractor boxes. Having leached off this preliminary cyanide solution as dry as possible, about the same quantity of strong solution, .25 or .3 per cent KCy, is then leached through the charge, after which the bottom discharge doors of this super-imposed vat are opened and the whole charge—now moistened with cyanide solution—is shoveled through the air into the treatment vats underneath, and carefully leveled off. The charge is then allowed to remain in contact for some hours with a second strong solution and this is followed by a .2 per cent or .25 per cent solution, and again by two or three weak washes from .15 per cent down to say .05 per cent in strength, after which a small water wash, say to the amount of 1 to 10 completes the treatment.

The whole cycle of operations generally takes six days, and the total amount of solution used is from 1 ton to 1½ tons per ton of tailings.

The residue sands are then discharged through the bottom

or side discharge doors into trucks or on to belt conveyors and deposited on the waste dump.

Not only have the mechanical devices been brought to a high state of perfection, but the consumption of cyanide has been reduced from the 3 or 4 pounds per ton of a few years ago to, say, 1-3 of a pound per ton of tailings, and to-day the total working costs on a modern cyanide plant may be put down at, say, 2/6 to 3/— per ton including redemption.

Slimes

Following quickly on the introduction of the cyanide process for the treatement of tailings or leachable sands the question of slimes treatment presented itself and this was undoubtedly one of the biggest problems which metallurgists and engineers were confronted with on these fields. Of the total ore crushed about 75 per cent only is leachable sands, the remaining 25 per cent being slimes—a product so fine that it is impossible to leach solutions through a charge. Attempts were made to treat slimes—which had been accumulated in a dam,—by first drying in the sun and then passing through a screen mixed with coarse tailings, so as to enable cyanide solutions to be leached through the charge, but this was not a success, and in spite of many other ingenious devices, such as keeping the charge agitated with compressed air introduced into the false bottom of the vat under the filter cloth, and in various other ways, the successful treatment of slimes was not accomplished until the now well known decantation process was introduced in 1895.

This process is carried out in two ways. If accumulated slimes from a dam is being treated, the product,—having been hauled from the dam in trucks,—is passed through a grizzly provided with cutters, into a mixing vat fitted with mechanical stirrers, and here the slimes are mixed with water and thoroughly disintegrated. Lime is added (to neutralize the acidity formed by the weathering of the pyrites) and also to assist the subsequent settlements of the slimes, according to the requirements which are carefully estimated.

From the mixing vat the charges flow or are pumped into another vat and allowed to settle. The clear water is then decanted off the surface and replaced by a weak cyanide solution, say 0.01 per cent KCy, and the whole charge thoroughly agitated either by means of mechanical stirrers fitted in the vats, or by centrifugal pumps, as explained later.

When the slimes have been agitated with this cyanide solution for from 6 to 12 hours, the charge is transferred to another vat and allowed to settle, the gold-bearing solution is then decanted off and further agitation, with very weak cyanide solutions, follow as in the treatment of current slimes direct from the mill, described hereafter.

The difficulties in the economic treatment of accumulated slimes on these fields are very great. In the first place the product is very low grade, and as a rule it has been stored for some years in paddocks or dams on the veld and has become contaminated with all kinds of organic matter. Owing to its heavy plastic nature it is difficult to handle, and it requires a large amount of lime to neutralize the acid compounds formed by partial weathering of the pyrites. The dissolving out of the gold is also difficult, owing to the lower oxygen-absorbing compounds of iron present and the extra cyanide these compounds consume, and again the excess of lime salts and other complicated agencies in the gold-bearing solution render the precipitation of the gold most difficult and erratic. However, in spite of all the serious obstacles encountered, accumulated slimes are now being treated on these fields at a handsome profit.

The treatment of current slimes from the mill is a far more simple matter, and since 1895, when the decantation process was first introduced here by Mr. J. R. Williams, large profits have been made from what was up till that time considered a worthless product.

Briefly, the manipulation of current slimes is as follows: The product (after its separation from the leachable sands by means of spitzkasten already referred to) flows down a long launder, in which a certain amount of lime is added, to a further series of spitzkasten and the action of the added lime is such that the slimes settle in the spitzkasten and are drawn off at the bottom, while the superfluous clear water flows away at the top. This thickened slimes pulp is run into a slimes collecting vat, where it is allowed to settle, after which the clear supernatant water is decanted off, leaving the product in the vat of a consistency of, say, 1 part of dry slimes to 1 part of water. The charge is now transferred (to another vat into which a quantity of cyanide solution has already been pumped), through the discharge pipe in the center of the concave bottom of the vat, which is connected with a centrifugal sludge pump, and while the transfer is being made the pulp is thinned down and sluiced out into the suction of the pump by a cyanide solution, played onto the mass through a hose fitted with a small nozzle. The strength of this solution is such as to bring the whole of the liquor in the charge up to 0.01 KCy. Having emptied the collecting vat, the suction of the sludge pump is now connected up with the bottom discharge pipe of the tank to which the charge has just been transferred, and an ideal agitation and circulation of the charge is effected by thus drawing off the pulp from the bottom of the vat and delivering it on to the surface again through the centrifugal pump. This circulation of the charge under cyanide solution is continued for, say, 1 or 2 hours, according to the requirements of the particular ore under treatment. The charge is then settled and clear solution decanted off. A second transfer of the charge, and subsequent circulation, follows under precisely similar conditions, with the exception that the cyanide solution is very much weaker—as low as .001 per cent KCy or weaker. The function of this solution is simply that of a wash by dilution so as to reduce the final liquor left in the slimes, in both gold contents and cyanide, as much as possible. In some cases, according to the richness of the slimes, it may be advisable to again transfer the charge with a water wash to further reduce the value of the solution held by the slimes, as it will be seen that the whole process is simply one of recovery by means of dilution.

The treatment having been completed, the residue slimes, containing, say, 50 per cent moisture, are now sent to the residue slimes dam either by sluicing out direct with water or by passing through the same centrifugal sludge pump used throughout the operations, which can be connected up to deliver into the discharge pipe or launder going to the slimes dam.

In treating 2½ dwts. to 3 dwts. current slimes an extraction of, say, 75 per cent is obtained; the total amount of solution used would amount to 5 or 6 tons per ton of dry slimes.

The total cost of treatment, including redemption, may be put down at from 3/— to 3/6 per ton, so it will be seen that even from the poor 2 dwts. slimes treated on many of the mines a handsome profit is made.

As slimes often contain a considerable quantity of sulphides, organic matter, or other compounds that require oxidation, it has been found advisable to use additional air, and this is very effectually accomplished by means of compressed air introduced into the delivery pipe during the circulation of the slimes through the pumps as described.

When treating low-grade slimes, to avoid undue loss of cyanide and over-taxing the gold extractor boxes with a greater bulk of solution than necessary, the practice is to "concentrate the solution."

The first solution decanted off goes direct to the solution clarifying and storage vat and then through the extractor boxes, and from the sump at the foot of the extractor box it is pumped up and used to transfer the settled slimes from the first treatment vat to the second, and from the second treatment vat the clear solution is pumped direct to transfer the untreated but settled charge of slimes from the collecting tank to

the first treatment vat, which thus completes the cycle. In this way the solutions are built up in strength and gold-bearing contents for the more efficient and economical extraction of their gold contents.

PRECIPITATION.

Now, with the introduction of the slimes process as described above, came the problem of the precipitation of gold from the necessarily very weak cyanide solutions employed—solutions too weak to yield up their gold by the ordinary zinc shavings method which is so effective with the stronger solutions obtained from the treatment of the leachable sands.

The successful precipitation of the gold from these very dilute cyanide solutions from the slimes plant was first accomplished by the Siemens & Halske electrolytic process which was introduced to these fields ten years ago, and after a very thorough and elaborate series of experiments the adaptability of the process to the work required of it was fully demonstrated and acknowledged to be a great improvement in many respects on the zinc process than is use.

The results obtained to-day by the Siemens & Halske boxes are about the same as when first introduced, but the great mechanical improvements which have been and are still being made have very considerably diminished both the initial and working costs of the plant. Ten years ago the working solutions used in the MacArthur-Forrest process varied from .25 per cent to .5 per cent KCy, and cyanide of potassium was then much dearer than it is to-day. The solutions used in the new electrical process were only .05 per cent to .15 per cent KCy, which thus made it possible to treat tailings and slimes of a very low grade value at a good profit. The claims of the two processes stimulated a healthy rivalry among metallurgists and engineers on both sides with the result that considerable improvements were made not only in both precipitation processes but also in the general arrangements of the entire plant.

At the May Consolidated Mine, where the electrolytic process was installed, a novel and most important change was made in the arrangement of the cyanide plant by building the collecting vats directly over the treatement vats, so that double treatment could be given to the ore at a minimum cost for transferring the charge from one vat to the other. This arrangement of super-imposed vats has since been adopted on all modern plants and has effected an enormous economy.

It will be interesting to follow the precipitation process from the start. In 1894, when the Siemens & Halske process was introduced, the position of its established rival was briefly as follows:

A 10 per cent royalty was charged for the use of the process.

Very strong solutions were used.

Precipitation of the gold from the solution was very good.

Consumption of zinc in the boxes was excessive and required replenishing constantly.

Clean up was complicated, tedious and wasteful of gold throughout.

An enormous quantity of rich slag was produced in smelting, no matter how well operations had been conducted.

Bullion bars were only 700 to 750 fine.

On the other hand the position of the Siemens & Halske process was:

A 2 per cent royalty charged.

Very weak cyanide solutions used.

Precipitation of gold from solutions not quite so good as in the zinc process.

Production of Prussian blue.

Once the boxes were in working order, very little attention was needed throughout the month.

Clean up extremely simple and little chance of any loss of gold.

Only a very small quantity of slags or skimmings of low value produced.

Bullion bars 950 fine.

It will be seen that the advantages at this time were in favor of the electrolytic process, and it was soon adopted by many companies who appreciated a process for which they only had to pay 2 per cent royalty, and which offered other advantages over the older process.

The new process would probably have been adopted by most of the mines had it not been for the fact that at this early stage of its development the MacArthur-Forrest patents in this country were upset, and thus the consideration of a 10 per cent royalty was no longer a factor in the case.

The keen rivalry which already existed between the advocates of both processes was now greatly accentuated and both sides worked hard for supremacy. Improvements to the zinc process rapidly followed, such as the treatment of gold slimes by sulphuric acid, the installation of filter presses to obtain the "clean up" in a fair shape for handling, smelting in reverberatory furnaces with clay-lined crucibles, etc.

The greatest drawback to the zinc process was the bad precipitation obtained from weak solutions, but this was also most successfully overcome by the introduction of the zinc lead couple which is accomplished by steeping the freshly cut zinc shavings in a weak solution (say 2 per cent of lead acetate). This couple, like several others which were tried, had been found fairly successful up to a certain point, but it was only when Mr. Betty hit upon the idea of adding a strong cyanide solution at the head of the precipitation boxes, that the activity of the zinc lead couple was fully exerted and gold sucessfully precipitated from the extremely weak solutions coming from the slimes plant.

The Siemens & Halske process was also greatly improved, but the now almost perfect precipitation obtained by the zinc-lead couple could not be obtained without incurring a big expenditure in the construction of the precipitation boxes and the advantages to be obtained would be very doubtful, owing to the increased amount of Prussian blue, which would be produced by the decomposition of the iron anodes added to which the extra amount of lead cathodes required would be out of proportion to the results obtained.

It will thus be seen that the position of the two processes to-day is briefly:

ZINC PROCESS.

No royalty.

Weak solutions.

Good precipitation.

Consumption of zinc still large and boxes require continual attention, and under certain circumstances give a lot of trouble, which sometimes result in very serious losses.

Clean up greatly improved and simplified, but still very bulky and requires a lot of handling.

Acid treatment and subsequent refining by manganese dioxide produces very fine bullion, which is also the case when the clean up is treated with litharge in a reverberatory furnace.

SIEMENS & HALSKE.

This process figures much about the same as has been already explained, although considerable economies and improvements in plant have been made here also.

Among the greatest drawbacks of this process the production of Prussian blue figured largely, also the amount of labor entailed in having to bag all the iron anodes. These bags, after being a certain time in the boxes, became clogged with Prussian blue, lime salts, etc., and greatly increased the resistance of the boxes, thus rendering it necessary to change the bags. These discarded bags constituted a by-product, which went by the name of "sacking," which contained a very considerable amount of gold. Four years ago, it was discovered that after an iron plate had been in the boxes a certain time the surface became evenly coated with iron oxide, and that this coating was sufficient to prevent the short circuiting between the iron anode and lead cathode.

If the iron plates are allowed to become oxidized before placing them in the boxes, the same results are obtained.

Bright patches on the plates are troublesome as they produce short circuiting very easily; the current will rush through these parts as they offer less resistance to its passage. With a little care, however, it does not take long to overcome this trouble, and when once the plates are in good working order no further trouble is experience with them throughout their life.

The great advantages the use of bare anodes offer are evident. The resistance of the boxes is greatly reduced, the Prussian blue formed finds its way to the bottom of he box and the trouble and expense of repeatingly bagging is avoided and there is no by-product in the form of sacking.

The precipitation of the precious metals upon the cathode is also much cleaner and brigher.

The results obtained from the working of the electrolytic process during the last year have been most creditable and compare very favorably with the best results obtained by the zinc process.

In 1903, 239,385 tons of sands or tailings were treated by the Siemens & Halske plants which produced 54,794 ounces fine gold. The highest actual extraction was 87.16 per cent, and the lowest 76.91 per cent from, say, a 4-dwt. product.

"Slimes" treated during same period amounted to 129,137 tons and produced 17,370 ounces fine gold. The highest actual extraction was 87.72 per cent, and the lowest 59.2 per cent. This last figure, however, was exceptional as old accumulated slimes of the worst and most contaminated description were being treated on this plant and the extraction here has since been improved to over 70 per cent.

The working of the Siemens & Halske process has been so often described in detail that it may appear perhaps somewhat of an unnecessary repetition to further describe it, but, by way of complimenting his paper we may say the iron anode and the lead strip cathode have proved the most practical and effective. The average density of current used is about 0.04 ampere per square foot of anode.

The anodes are placed from 3½ inches to 5 inches apart, and the cathodes, consisting of three or four sheets of thin lead, cut into strips about 1 inch wide, and suspended from a two or three wire frame, are placed between the anodes, care being taken that they hang free of the iron anodes.

The size of the box, which has proved most efficient, is about 30 feet long, 4 feet 6 inches to 6 feet wide, and 5 feet deep. The compartments are about 2 feet 2 inches wide, and allow the easy placing of bunches of lead strips 4 feet long by 2 feet wide. The solution is circulated through one compartment into the next in an upward direction as in the ordinary zinc boxes.

Six boxes of this description will easily deal with all the solutions coming from a sands plant on the basis of 100 stamps crushing 5 tons a day.

The value of the gold contents carried in the solution entering the first two boxes, called the strong boxes, will in no material way influence the amount of gold carried off by the weakest solution after going through the last boxes, called weak boxes. That is to say, that no matter whether the solution entering the boxes carries 4, 10 or 12 dwts. of gold per ton, we find the solution leaving the box and going to weak storage does not increase proportionately to the original value of the solution, and it will always carry from .5 dwts. to a dwt. of gold per ton.

It is this apparently irreducible minimum that is the greatest drawback to the electrolytic process, and has retarded its progress to a great extent.

The following details obtained from one of the plants now working this process may be taken as a fair criterion of the work done all round.

Capacity of plant 8,000 tons of sands and coarse spitzlutten concentartes.

Precipitation Plant.

Six boxes, 30 feet long by 5 feet 6 inches wide by 5 feet deep, divided into six compartments, each 4 feet by 4 feet 8 inches.

Number of anodes (iron), thirteen, size 4 feet by 4 feet.

Number of cathodes (lead strips) twelve, size 4 feet by 2 feet (two frames, 2 feet by 4 feet, between each set of anodes).

Sheets of lead in each frame, three (2 feet by 4 feet size, weight 1 pound per sheet).

Amperes per box 125, volts 1.75.

The compartments of the boxes are coupled in parallel, and, each box or boxes, again in series with the next set, and so on. The total current used is 250 amperes, and e. m. f. 10.5 volts.

The flow of gold-bearing solution through the boxes is at the rate of 50 tons to 75 tons per 24 hours per box, and results are as follows:

Strong box—One, 50 tons per 24 hours. Solution entering boxes 6.4 dwts. gold per ton. Solution leaving box 1.63 dwts. gold per ton, = 74.5 per cent extraction.

Medium boxes—Three, 150 tons per 24 hours. Solution entering boxes, 5.36 dwts. gold per ton. Solution leaving boxes, 1.82 dwts. gold per ton, = 66 per cent extraction.

Weak boxes—Two, 150 tons per 24 hours. Solution entering boxes, 2.36 dwts. gold per ton. Solution leaving boxes, 1.32 dwts. gold per ton, = 44 per cent extraction.

An improvement on these results, from purely an extraction point of view, could, of course, be obtained by decreasing the rate of flow, which would mean the erection of more boxes as the quantity of solution used could not be decreased economically. The foregoing remarks, with reference to solutions from "sands" treatment, may be, generally speaking, applied to solutions from the slimes plant, due allowance being made for the difference in gold contents. Solutions from the slimes plant generally run from 1 dwt. to 2 dwts. per ton, and to precipitate them down to say 4 grains, leaving the boxes it would require about 8,000 square feet of active anode surface.

Ever since the introduction of the Siemens & Halske process now ten years ago, unceasing and strenuous efforts have been made to try and reduce the value of solutions leaving the boxes, as low, or lower, if possible, than the results now obtained by the zinc process which average, say, .09 dwts. per ton of solution. Eminent men, such as Professor von Oettingen, of the Leipzig University, have been consulted in the matter, but the solutions are still leaving the boxes with a dwt. of gold and the problem has yet to be solved.

Comparisons with the results obtained by this process in other countries are of little use, as the circumstances are so different and present an entirely different problem to that obtaining on these fields.

For instance, on a plant where the solutions entering the boxes carry from 15 dwts. to 30 dwts. of gold, 8 to 10 ounces of silver and 1 per cent to 2 per cent copper, it is considered highly satisfactory, if the solutions leaving the boxes do not exceed 3 dwts. of gold, 0.5 to 1 ounce silver and 0.5 per cent copper. Electrical processes are responsible for the only really satisfactory work done with such solutions as the above.

On some of the plants on the Rand the solutions entering the boxes only carry 3 dwts. gold, the silver and copper being negligible quantities.

A few years ago Andreoli obtained excellent results in London on an experimental electrolytic plant, and it was hoped that his process would prove as successful here as it did in London, but in practice it failed to improve matters with our working solutions. Briefly, the theory of Andreoli's process consists in using a peroxidized lead anode and an iron cathode. After the cathode is considered to be sufficiently coated with gold it is plunged into a bath of molten lead; the lead alloys with the gold, and after all the cathodes have been treated in this manner the lead bullion is cupelled.

After perusal of these details on the working of electrolytic processes, it will be seen that the field is still open to the

electrometallurgist. The ideal process sought is one that will precipitate gold from its solution down to, say, 4 grains per ton with a moderate size plant, will produce no by-products, and the bullion bar must be obtained in a simple manner and of a high standard of fineness.

That the Siemens & Halske process has been and is a very important factor in advancing the metallurgical practices on the Rand, both directly and indirectly, cannot be gainsaid. Electrochemistry is a scinece which is making rapid advancement and the day is probably not far distant when we shall see the imperfections of such processes as the Siemens & Halske overcome; even now the variations in costs and results between this and the zinc process are very slight, and, although at present the latter process shows a slightly better average in precipitation, the opportunities for losing gold throughout the various operations, before eventually producing a lead bullion for cupelling (which is obtained first hand in the electrolytic method) must go a long way towards counterbalancing its advantages under this head.

By-Products.

At the end of nearly, if not all, metallurgical operations, we find a product left which still contains a certain amount of the metal sought which cannot be recovered without special treatment. These materials are conveniently called by-products and the value of the original metal being exploited which they still contain, is the determining factor as to their ultimate disposal. These by-products can only be treated successfully in specially constructed plants, and as the individual mines do not produce by-products in sufficient quantities to warrant each in erecting special plants, the custom has been to accumulate the material for a time and then either ship it to the big metallurgical centers abroad or sell it to a local reduction works company.

The Rand Central Ore Reduction Co. was established here some twelve years ago, and erected very fine works for the special treatment of every kind of by-products produced on these fields and the majority of the gold mining companies availed themselves of these works for the disposal of all by-products.

In the early days here before the introduction of the cyanide process the by-products produced consisted principally of concentrates, black sands, sweepings, battery chips, and screenings, slags from the smelting of battery gold, old crucibles and ashes. Of course "tailings" and "slimes" also constituted a by-product in those days, but we have already described how the enormous accumulation of these materials, once regarded as nothing more than a nuisance, has since been turned into such a huge source of revenue through the cyanide process.

The working of this process is responsible for the production of most of the by-products which now have to be dealt with on these fields, such as slags, white slimes, Prussian blue, scrapings, sweepings, skimmings, dross, litharge, brick dust, test bottoms, old crucibles and liners, ashes, sump sediments, etc.

If the gold has been precipitated on zinc the subsequent smelting of the gold slimes results in the production of a considerable quantity of slags, containing more or less metallic gold distributed throughout the mass in the form of shotty prills, in which form, after grinding the slags, it can be recovered by cradling, but a large amount of gold remains which is so thoroughly assimilated by the slag itself as to render its extraction impossible by ordinary means.

As an example of the enormous amount of slag sometimes produced, in the days of smelting with ordinary plumbago crucibles in an ordinary furnace, we may mention that the proportion has been as high as 6 tons of slags for the recovery of about 2,000 ounces of gold.

On one of the mines every endeavor was made to obtain as high an extraction as possible from the slags by crushing them with water through a 700-mesh screen and amalgamating over specially prepared copper plates. A considerable amount of gold was recovered in this way after which the residue pulp was carefully collected in settling vats, placed below the copper plates, and this residue still contained from 10 to 12 ounces gold per ton. The residue slag pulp accumulated over a period of 7 years by this company was subsequently sold to the Rand Central Ore Reduction Company and realized about £12,000.

If the gold from the cyanide solutions has been precipitated electrolytically on lead sheets, the lead bullion obtained carries nearly all the gold, and this is readily obtained in a very pure state by cupellation. The by-products produced by this method are, skimmings, metallics from the smelting operations, and test bottoms from the cupellation. The metallics can be added to the lead when cupelling, but the remainder require special treatment at the smelting works. The "clean up" process from the zinc plant has fortunately been greatly improved. The first step was the introduction of reverberatory furnaces in which one or two dozen crucibles could be placed at the same time and their contents smelted at a uniform heat, this lessened the smelting operations considerably but the quantity of slag produced and crucibles used remained as before.

The next step forward was the treatment of the zinc clean up with sulphuric acid—special lead-lined clean up vats being used for this purpose.

These acid vats were covered over with a conical hood dipping in a peripheral launder filled with water which hermetically closed the whole apparatus—a flue or pipe fitted into the apex of the conical hood and conducted through the roof of the building carried off the gases produced in the vat.

After washing out all the soluble salts the gold slimes are pumped into a filter press and the resulting cakes dried, fluxed, and smelted.

By this means the amount of slag produced is greately diminished but it still carries a considerable amount of gold which can only be recovered by some special process.

The bullion produced by the old method of smelting with borax, soda, sand, etc., was of a low grade, from, say, 650 to 750 fine only, but by the use of manganese peroxide in the flux a very high grade bullion is obtained, the only drawback to this flux is that while it eliminates the base metals it also carries the silver into the slag.

The latest improvement in the smelting operations, known as the Tavaner process, consists of placing the acid treated zinc gold slimes in a reverberatory furnace with litharge and slag from the laboratory or any other convenient flux and a reducing agent.

This brings the precious metals into lead bullion, which is cupelled and yields a bullion of very high standard.

The slag produced by this method will be of very low grade value if the operations have been carefully conducted. The drawback to this process at present is the large amount of gold which remains locked up in the furnace, and the somewhat heavy cost for maintenance—nevertheless, it is a process working in the right direction, and has done a great deal towards enabling the individual mines to treat the various by-products on the spot, instead of selling to customs works.

Most of the mining companies here had hitherto sold their by-products to the Rand Central Ore Reduction Co.

For the last eight years this company has been treating by-products very successfully in pan and blast furnaces. According to the gold contents and nature of the by-product it was sent to either one or other of these furnaces.

Pan Furnace.

Slags from the zinc clean up, and any product assaying over 30 ounces gold per ton, were treated in the "pan" furnace.

The slag is fed into this furnace with a certain quantity of litharge and a reducing agent, such as coal or ground plumbago crucibles, the other by-products being added in

such quantities as would not interfere in obtaining a fluid slag. The litharge on being reduced combines with the precious metals contained in the charge and accumulates at the bottom of the furnace as lead bullion; the slag is drawn off as often as necessary and when the lead bullion has accumulated in sufficient quantity at the bottom it is also tapped off into moulds, and may be immediately cupelled and a high-grade bullion obtained. The disadvantages of the furnace were: the heavy wear and tear, and the large amount of gold which remained locked up in it, and which could not be recovered until the furnace was completely broken down; the gold left in the resulting slag generally amounted to from 2 to 4 ounces per ton, and the quantity of lead was also considerable, amounting to 8 to 10 per cent.

Blast Furnace.

By-products which for any reaspn could not be successfuly treated in the pan furnace were dealt with in a "blast furnace."

The principal fluxes used in this furnace were iron oxide and carbonate of lime; litharge and galena produced the lead which acted as carrier for the precious metals; and sulphurets served to produce a matte with whatever copper was present.

This blast furnace enabled by-products of any description to be smelted, but the high price paid here for the coke fuel and necessary fluxes rendered the work very expensive. For every ton of by-product, from 3 to 4 tons of fluxes and fuel had to be smelted, which considerably reduced the margin of profit.

The slags produced by this furnace seldom carried more than a few pennyweights of gold and silver or more than 1 per cent of lead, if of higher value they were repassed through the furnace.

The lead produced, which carries the precious metals and a small quantity of copper, was treated according to requirements. If the flux galena was abundant, the lead was deprived of the contained precious metals by the Parkes process.

Parkes Process.

This, briefly, consists of smelting the lead bullion bars obtained from the blast furnace in cast iron pots of about 20 tons capacity. The copper dross is skimmed off and zinc is then added to the mass of molten metal.

Zinc has the extraordinary property of seizing nearly all the gold and silver present, and bringing it to the surface in the form of a crust which is skimmed off. This scum—which consists of zinc, lead and precious metals—is sent to the pan furnace for the purpose of fluxing the zinc. When this has been done the lead bullion produced is ready to be cupelled at once. The zinc may also be distilled and the remaining lead cupelled to obtain the gold and silver. If galena is difficult to obtain, then the lead as it comes from the blast furnace is cupelled and the litharge so produced used again in the blast furnace as already described.

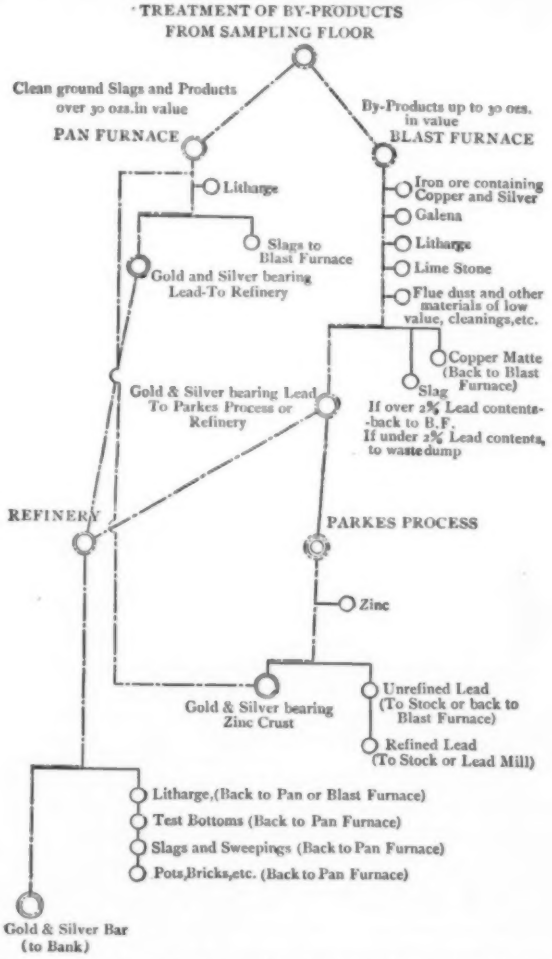
In cases where the use of galena results in a great excess of lead bullion, this is refined and converted into marketable lead after the precious metals have been withdrawn.

In connection with the blast furnace work a fair amount of flux dust is produced. This necessitates the regular cleaning of the flues and dust chambers where it collects. The collected dust is simply added to the charges and returned to the blast furnace.

The copper matte is also passed through the furnace with the object of concentrating it, and when its contents are considered sufficiently high, is shipped to Europe for further treatment.

All the refining which has been carried out at these works has merely consisted in the separation by cupellation of the gold and silver from the lead and other base metals—so far as can be done in this operation. The bullion bars of gold and silver have been sent to Europe for further treatment and disposal.

A graphic table is appended which will show at a glance the course of the by-products, fluxes, and secondary by-products, and will, we hope, render the foregoing pages relating to the Rand Central Ore Co.'s plant to be more easily understood.


TREATMENT OF BY-PRODUCTS FROM SAMPLING FLOOR
Clean ground Slags and Products over 30 ozs.in value
By-Products up to 30 ozs. in value
PAN FURNACE
BLAST FURNACE
Litharge
Iron ore containing Copper and Silver
Galena
Litharge
Lime Stone
Flue dust and other materials of low value, cleanings,etc.
Slags to Blast Furnace
Gold and Silver bearing Lead-To Refinery
Copper Matte (Back to Blast Furnace)
Slag
If over 2% Lead contents -back to B.F.
If under 2% Lead contents, to waste dump
Gold & Silver bearing Lead To Parkes Process or Refinery
REFINERY
PARKES PROCESS
Zinc
Gold & Silver bearing Zinc Crust
Unrefined Lead (To Stock or back to Blast Furnace)
Refined Lead (To Stock or Lead Mill)
Litharge,(Back to Pan or Blast Furnace)
Test Bottoms (Back to Pan Furnace)
Slags and Sweepings (Back to Pan Furnace)
Pots,Bricks,etc. (Back to Pan Furnace)
Gold & Silver Bar (to Bank)

A paper recently read by one of the authors before the South African Association for the Advancement of Science on the "Evolution of the Treatment of By-Products on the Rand, the "Evolution of the Treatment of By-Products on the Rand."

In conclusion, although great advances have been made in the metallurgy of gold on the Rand, plenty of problems still await solution. Here money is generously, if not lavishly, spent if there seems any chance of effecting improvements, the friendly rivalry, coupled with the exceptional facilities which exist for the interchange of ideas and experiences, all help in the same direction, and this is one of the most pleasing and notable features of the scientific life on the Rand.

Johannesburg, South Africa, April 18, 1904.

American Institute of Electrical Engineers.—The annual meeting of this Institute was held in New York City at the Chemists' Club on May 17 and 18. It had the same experience as the American Electrochemical Society at its recent annual meeting in Washington, namely, that the attendance was comparatively small, probably on account of the coming International Electrical Congress in St. Louis. With the exception of a paper of Mr. W. E. Winship on the Calculation of "Floating" Batteries, no paper of special interest to electrochemical engineers was presented.

ELECTROLYTIC REFINING OF GOLD.

BY EMIL WOHLWILL, PH. D.

At the request of the editor of ELECTROCHEMICAL INDUSTRY, I herewith give some notes on the electrolytic method of gold refining, to supplement the article of Dr. Tuttle (Vol. 1, page 157, January, 1903). My electrolytic method of refining gold was introduced by Dr. D. K. Tuttle into the United States Mint of Philadelphia, and I am pleased that, after two years of use, he calls it "a great boon for the patrons of the United States Mint." This method was the result of experiments which were undertaken by me for the single purpose of investigating whether in parting platinum from gold the usual method can be replaced by an electric method. These experiments were made by me as director of the Deutsche Affinerie in Hamburg, and were begun in 1874. I established the following facts, which were unknown up to that date:

(1) In an electrolyte of chloride of gold which contains no other chloride, a pure gold anode, or an anode rich in gold, behaves like an anode of platinum or carbon. Chlorine is developed in gaseous form, while the gold is not dissolved.

(2) This development of chlorine gas can be prevented and gold is dissolved from the anode, if hydrochloric acid or certain other chlorides are added to the electrolyte of chloride of gold.

(3) There exists for a given solution a maximum of current density at which gold is dissolved from the anode without a development of chlorine gas. If the current density is increased beyond this maximum, chlorine gas is developed. However, by increasing the quantity of hydrochloric acid, or chlorides which act in the same way, and by increasing the temperature of the electrolyte, this maximum of the current density is increased.

GOLD REFINING ROOM OF THE DEUTSCHE AFFINERIE IN HAMBURG.

(4) In a solution of chloride of gold, even with the addition of hydrochloric acid or of such chlorides which have the same effect, and even with increased temperature, pure platinum is not dissolved from the anode, and causes, therefore, the development of chlorine gas. If, however, platinum is contained in an alloy with gold, and such an alloy is used as anode in an electrolyte of chloride of gold with an addition of hydrochloric acid, the platinum goes into the solution in the same manner as the gold, while at the cathode gold is precipitated, completely free from platinum, from the solution containing gold and platinum; the platinum remains in the solution.

These facts represent the principles upon which the method is based by which, since 1878, the Norddeutsche Affinerie in Hamburg produces chemically pure gold from impure gold, and especially from gold containing platinum. The facts given above also indicate a very simple way by which from gold containing platinum the smallest quantities of platinum can be completely won. Since platinum is dissolved from the gold anode, but is not precipitated from the solution, it is possible to collect the platinum from a great many impure gold anodes in the same volume of solution and to continue with this process until the concentration of the platinum in the solution is sufficiently high to permit direct precipitation by chloride of ammonium.

Electrolytic gold, produced by this new method from a material rich in platinum, was first exhibited at the Paris Electrical Exposition, in 1881, in form of the original cathodes as well as in form of bars which had been obtained by melting the former. At the request of M. Debray, at that time director of the Mint of Paris, the exhibited gold was subjected to an analysis. The test certificate of the Paris Mint, which I still have, records a purity of 1,000 in 1,000; M. Debray remarked that this is a purity which had not been obtained before.

The method as it was used at that time had an essential disadvantage besides its great advantages. It was the cleanest operation imaginable; any quantity of gold was dissolved in uncovered vessels, without producing any smell of chlorine. Even an evaporation of hydrochloric acid from the hot, but only slightly acidified solutions was scarcely noticeable. A higher price was paid for the precipitated gold on account of its purity. The operation, however, was very slow, much slower than the usual chemical method (solution by means of aqua regia and precipitation of the gold by protochloride of iron). With the chemical method the total contents of gold could be obtained in twenty-four hours from the largest quantities of gold containing platinum. On the other hand, the electrolytic process required generally three times twenty-four hours to obtain only about 80 per cent of the contents of gold as first product. This was evidently a very distinct disadvantage.

In order to reduce the electrolytic method to a really industrial process, it was therefore absolutely necessary to increase the speed of the operation. To a limited extent the speed could be increased by using anode plates of smaller thickness, but a really satisfactory reduction of the time of operation required a considerable increase of the current density, but this appeared to be impossible beyond a certain limit. When the current density was increased up to a certain limit, any further increase, either by increasing the current with constant anode surface or by diminishing the anode surface with constant current, resulted in the evolution of chlorine gas at the anode, and the quantity of gold dissolved from the anode was therefore smaller than that of the gold precipitated on the cathode.

The remedy was found in a consequent application of the principle indicated above under (3). On the basis of the facts stated there I had given for many years the following rule to my workingman: When you smell chlorine, measure at once the temperature; if the thermometer shows less than 67° C., heat the bath; if this is not sufficient to make the smell of chlorine disappear, add hydrochloric acid in small portions

to the solution. At that time, as long as the increase of speed of operation was not absolutely necessary, I kept the average addition of hydrochloric acid as small as possible, and, as a result, I regarded a relatively low current density as normal. However, when I seriously asked myself the question whether the arbitrarily fixed limit represented really the normal condition (i. e., the limit attainable on account of the principle of the method) I soon discovered that this was by no means the case.

Experiments proved that by continually increasing the amount of hydrochloric acid in the electrolyte, continuous solution of the gold anode without any evolution of chlorine gas could be obtained at current densities which were six times greater than those which I formerly considered as normal; but less than one-half was sufficient for practical purposes, i. e., for an operation in which from any quantity of impure gold, 80 to 90 per cent of the total contents of gold are precipitated in the morning, if the plates had been placed into the baths the afternoon before. Only in this improved form the electrolytic method was able to completely replace the chemical method which up to that time had been used exclusively for the same purpose. That the substitution of the electrolytic for the chemical method was advantageous has been recognized first by the largest German parting plant, the Deutsche Gold und Silber-Scheideanstalt (vormals Roessler), in Frankfort-on-the-Main, many years after the introduction of the electrolytic method by the Norddeutsche Affinerie. In the Frankfurt plant the chemical method had been used for a long time for parting platinum from gold, it had been developed there to great perfectness, and had been employed on the largest scale. Since 1896, however, this chemical method was given up, and the electrolytic method was introduced instead, and the latter has since been used continually for refining of gold won in the plant.

Dr. Tuttle, in his article mentioned before, also calls attention to the very simple method of recovering all platinum contained in the gold as one of the advantages of the electrolytic method.

For the same main purpose, *i. e.*, for refining gold containing platinum, the Royal plant of Freiberg-Halsbrücke has installed in May, 1903, a smaller electrolytic plant, in which four months after erection more than 6 kg. of pure platinum had already been won.

It seems not superfluous to discuss somewhat more in detail the question how far a method for completely parting platinum and gold can be of practical importance.

Wherever exists an industry of noble metals the scrap from such industries represents an important object for the work of the parting plants. In this scrap a certain, not negligible, content of platinum will be present, for the reason that the manufacture of vessels which consist entirely or partly of platinum, or which are covered with platinum, takes place often at the same places as the manufacture of gold and silver articles. As a result, scrap will contain the three metals in various quantities. In view of the high price of platinum every well-conducted parting plant must endeavor to win the total content of platinum, besides gold and silver from the mixed scrap. This will be especially necessary if the parting plant has to pay for the platinum.

But this more incidental and irregular occurrence of platinum does not represent the richest of the sources from which platinum comes into the plants devoted to the parting of gold and silver. It is well known that in nature at many places gold and platinum occur together. In this way platinum contained in gold, together with its kindred metals, iridium, palladium, rhodium, etc., is supplied from the gold mines to the parting plant. Less attention appears to have been paid to the fact that even at places where platinum, iridium, etc., are not found separately, the gold and silver found in nature have been found to contain platinum.

More than fifty years ago Max Pettenkofer discovered the content of platinum in the noble metals, and stated that "all silver which is found in trade and which does not come directly from a parting plant, contains platinum, so that this metal, which has been considered as extraordinarily rare, can be obtained in larger and profitable quantities if the silver is subjected to a process in which the platinum remains in the residue, that is, on a large scale in the treatment with sulphuric acid and in amalgamation." Pettenkofer would add to-day, "and in the electrolytic method of parting silver containing gold."

As far as I know quantitative determinations of this contents of platinum in silver, or in gold, won from silver, have never been published, and in the literature I cannot even find notes which would confirm the statement of Pettenkofer that all silver which occurs in nature contains platinum. It should, therefore, be of interest to give here some data for which I am obliged to Mr. Ludwig Opificius, chief chemist of the Deutsche Gold-und-Silber-Scheideanstalt in Frankfurt.

At the time when a uniform system of money was introduced in Germany the largest part of the old German coins were melted and the melted metal was subjected to a parting operation. At that time Mr. Opificius tested the various kinds of silver coins and also several sorts of silver obtained from metallurgical plants, and determined the content of platinum in quantities of silver between 5 and 10 kg. The results obtained by him are given in Table I. (I add for the greater convenience of comparison figures for the content of gold which was determined simultaneously from smaller quantities, as far as they have been placed at my disposal by Mr. Opificius):

TABLE I.

			Gr. gold.	Gr. platinum.	Gr. platinum per 1 Gr. Gold.
10	kilogram	French Laubthaler.....	...	0.016	.....
5	"	French Laubthaler.....	...	0.009	.....
5	"	Spanish Säulenthaler.....	2.8	0.011	0.0039
10	"	Coins, each 1/6 thaler.....	4.5	0.025	0.0055
10	"	Coins, each 1/6 thaler.....	4.5	0.0225	0.0050
10	"	Coins, each 1/3 thaler.....	2 to 3	0.0275	0.0110
10	"	Thaler	...	0.0145	.....
10	"	Thaler	...	0.0245	.....
10	"	Prussian Thaler..... (from 1820 to 22)	6.2	0.016	0.0026
10	"	Conventionsthaler	6.5	0.0165	0.0025
10	"	Conventionsthaler	6.5	0.0180	0.0028
10	"	Kronthaler	7.6	0.0250	0.0033
10	"	Kronthaler	7.6	0.0245	0.0032
10	"	Coins, each 20 kr. (Austria)	1.5	0.0135	0.0090
10	"	Coins, each 20 kr. (Austria)	4.8	0.0220	0.0046
10	"	Iambos .....2.2 to 4.7		0.0300	0.0136—0.0078
10	"	Iambos	...	0.0365	.....
10	"	Emser Silber	4 to 6	0.029	0.0058
10	"	Emser Silber	...	0.029	.....
10	"	Eschweiler Silber.....	5 to 10	0.0482	0.0096—0.0048
10	"	Eschweiler Silber	...	0.0477	.....

The average of sixteen determinations in Table I. shows, therefore, that gold parted from silver by means of sulphuric acid or electrolysis contains 6 milligrams of platinum per one gram of gold. This corresponds to a value of about $1.co for each 10 ounces of gold.

Mr. Opificius has not published these figures, because in dissolving the silver in his experiments, concentrated sulphuric acid was used. According to investigations made at a later date this acid can never be considered as free from platinum if it was concentrated in platinum vessels. It is, therefore, possible that a part of the platinum which was found came from the acids; its amount cannot be estimated; the same suspicion might also be raised against the observations of Pettenkofer. No investigation has as yet been made whether in parting silver by means of sulphuric acid containing platinum, platinum is really precipitated.

I do not think it is probable that the quantity of platinum which in this way is added to the original content of platinum in the noble metals is an appreciable amount. In the Norddeutsche Affinerie, in Hamburg, gold and silver were parted exclusively by means of sulphuric acid up to 1894; in the next years this method was replaced by the electrolytic process with the use of a nitrate solution, first only to a small degree, later on nearly completely. The material of the parting operation was essentially the same before and after 1894; all

gold which was obtained was afterwards treated to win the platinum from it. If a considerable amount of the platinum obtained in this way had come from the use of sulphuric acid the total quantity of platinum recovered should have decreased in considerable degree as a consequence of the introduction of the electrolytic process. From the record books of the Norddeutsche Affinerie, however, such an influence has not been found at all. The figures of Mr. Opificius may, therefore, be considered as not completely certain for the circumstances mentioned, but there is no reason why they should not be considered as approximate figures for the real content of platinum in the tested sorts of silver.

These figures relate, as can be easily seen, to silver with a relatively small content of gold, which was not parted at that time since the small quantity of gold would not have paid for the cost of the operation of parting by the method in use at that time. The investigations of Pettenkofer also appear to refer mainly to silver poor in gold. From the experience of the German parting plants, however, there is no doubt that the content of platinum is not limited to such sorts of silver poor in gold. It seems very probable that American silver is no exception from this general rule. Up to the present I have tested only a few products of American silver parting plants, but have found in them platinum without any exception, and in some a relatively high content of platinum.

Considering the large quantities of noble metals in various mixtures which are produced every year in the United States, we may conclude from the facts mentioned that very considerable quantities of platinum are supplied to the works which part gold containing silver or silver containing gold. In view of the present market price of platinum we must assume that the income of the producer and refiner would be increased by a considerable amount if the platinum would be extracted from the gold in which it is contained. One more reason why this extraction should be carried out is that while for his main purpose the refiner may not need to undertake the recovery of platinum, yet he cannot avoid to separate this substance if it is present in his gold. Even before it was known that platinum exists in pure gold, refiners used to remove the unknown impurity by repeatedly melting with potassium nitrate, and the same method is to-day used by those who want to get as pure gold as possible without parting the platinum from it by a special process.

The presence of platinum in any considerable quantity results in a certain quantity of silver being retained in the gold in such a way that it cannot be removed by the ordinary methods of parting. Moreover, where there is platinum there is very often also a certain addition of iridium, which, even in very small quantities, renders the gold brittle. These disadvantages can, indeed, be overcome by melting with potassium nitrate. One obtains by this method nearly pure gold, since platinum and iridium with a small quantity of gold pass into the slag, from which the platinum may then be obtained in a somewhat roundabout way, which has already been described by Pettenkofer. It may be doubted that this method is used to any extent in the United States, either by the producers of gold or by the parting plants to which the slag is sold. It is at least certain that scrap of various kind is exported every year abroad containing, besides gold, very considerable quantities of platinum.

It seems superfluous to remark that in comparison with a process by which, as described, an obnoxious although valuable impurity is simply removed, another process is preferable by which the valuable impurity is not merely removed, but recovered in a simple way. If we apply this general remark to the refining of gold it means that the rational solution of the parting problem must consist in a process by which the gold is obtained in an entirely pure condition, while the total platinum which was present in it is also recovered.

For this purpose, since Pettenkofer's discovery, the repeatedly mentioned chemical method has been in use which had also been suggested by Pettenkofer. The impure gold is dissolved in aqua regia, and from the dilute solution the gold is precipitated by protochloride of iron, free from platinum, etc. The solution from which the gold has thus been removed, is treated with metallic iron, and all platinum is thereby precipitated, first mixed with most of the other impurities of the gold. The advantages of the electric process, in comparison with this method, are of a manifold nature.

Whoever has been accustomed to dissolve larger quantities of gold every day in a mixture of hydrochloric and nitric acid, making use of more or less good ventilation, will scarcely consider it a disadvantage that in this process it is impossible to avoid the enormous development of gases and vapors of horrible smell and detrimental to health. Nevertheless, I have not yet found anybody who was accustomed to this disadvantage and who was not pleasantly surprised when entering the working room of a gold electrolytic refining plant and found that here a continuous solution of gold took place under the action of the current in open baths without the slightest development of smelling gas. It may be claimed that in our days the possibility of replacing an industrial process by another one which is better in hygienic respects, and has no other disadvantages, represents a serious consideration in favor of the latter. Of greater importance, however, will be the proof that the new method is also superior in diminishing the cost. I believe that the electric method of gold refining fulfills this condition.

The consumption of acid is more than fifty times greater with the chemical method, and against this main item of cost there is, on the side of the electrolytic process, only the cost of electric energy, of which a very small amount (about 300 watt-hours per kg. gold) is sufficient. For wages an amount is to be charged which is certainly not higher than for the work with aqua regia, especially since in accordance with the nature of the process, the work is not interrupted during the night, but is so arranged that that part of the electrolytic action which needs no attendance whatever, goes on during the night hours; for the manual work which is to be done during the day hours not more than one workingman is required for the treatment of 50 kg. (of rather pure gold). An estimate of the wages will, therefore, give a result in favor of the electric process even if a proper interest is charged for the continual stock of gold which is required for the electrolytic process, and if an extra charge is made for melting the scrap of the anodes and of fine gold serving for the cathodes.

Of the other advantages of the electrolytic process I briefly mention the greater chemical purity of the obtained gold, the essentially greater simplicity of recovering the platinum, and the complete separation of platinum from iridium and other platinum metals.

Platinum can be obtained directly by the chemical process only if the content of platinum in the gold is very high; it is the more complicated the smaller the quantity of the platinum in the gold. In the electrolytic process very small or very large quantities of platinum in the gold anodes pass into the electrolyte with equal facility, and the production of platinum from the electrolyte is always direct.

By the treatment with aqua regia a part of the iridium is dissolved together with platinum, iridium being nearly always present where there is platinum. From such a solution containing iridium, the platinum is then precipitated mixed with iridium if one does not use a specially complicated method for parting the two metals. On the other hand, platinum which has accumulated in the electrolyte, and which is obtained from it by precipitation, is absolutely free from iridium, because the latter is not dissolved from the anode by the action of the current. Iridium and osmium-iridium, which are present in gold, accumulate therefore completely and free from platinum in the anode sludge and may easily be obtained from the latter.

After all that has been said, the electrolytic process of refining gold appears to be especially suitable for overcoming a deficiency in the chemical and electrochemical industries of the United States; this deficiency is the non-winning of platinum present in gold. The production of platinum, as recorded by American statistics, certainly does not differ very much from a complete lack of production of platinum.

According to the statistics in Rothwell's *Mineral Industry*, the total production of platinum in the United States during eight years was as follows:

1895	150	ounces.
1896	162	"
1897	200	"
1898	300	"
1899	0	"
1900	173	"
1901	1408	"
1902	94	"

If we neglect the exceptionally high output of 1901, the average production was 154 ounces per year. The output of the year 1902 is the lowest on record since the statistical data on the production of platinum in the United States were undertaken. That the above quantities of platinum are parted from gold is said on page 569 of volume 7 of *Mineral Industry*. But the relatively high quantities of iridium which are mentioned side by side with platinum, justify the supposition that the produced platinum was not the result of a real refining process of gold, but came from the treatment of a mixture of gold with platinum ore. On the other hand, there cannot be any doubt that such refining processes are carried out in the United States on a very large scale, and cannot be carried out, or, at least, will not be complete without the separation of platinum. It would be interesting to find out what has become even only of those quantities of platinum which were won together with gold in the anode sludge, since the introduction of the electrolytic treatment of silver, and which were parted from the gold during the treatment of the sludge.

What has happened to platinum during this time has also happened to the corresponding quantity of iridium, since in gold containing platinum, iridium is nearly always present.

Much greater is probably the amount of palladium which was obtained at the silver anode of the Moebius refineries, but was not won from the sludge. It is known that in the Sudbury district in Ontario copper ores containing nickel are won and treated which also contain platinum and palladium in no small quantities, and that in these ores the content of palladium is very near to that of platinum. It was probably raw copper produced from these ores which was treated more than twenty years ago in large quantities in Hamburg under my direction. By electrolytic treatment of the copper, an anode sludge was obtained which was surprisingly rich in palladium. Together with the platinum which was also present, it was collected by a metallurgical process in silver containing gold; it was separated from the silver together with gold; as part of the gold anodes it passed into solution during electrolysis and from the solution it was completely recovered in pure form.

There is good reason to believe that the Sudbury ores are not the only ones in America in which the quantity of palladium present together with platinum is considerable. I do not know the price of palladium in the United States, but I know that one of the most prominent platinum factories of Germany pays at present 2,000 marks ($476) for one kilogram of palladium. It is therefore possible that if palladium is present, the value of the material thrown away with the ordinary parting method is very considerable. This amount could be easily recovered by the use of the electrolytic method.

In a second article the electrolytic separation of gold from gold alloys will be dealt with.

Hamburg, Germany, May 5, 1904.

ANALYSIS OF REFINED COPPER.

In our March issue, pages 87 to 95, we published a paper on "a laboratory study of the stages in the refining of copper," presented by Messrs. H. O. Hofman, C. F. Green and R. B. Yerxa to the American Institute of Mining Engineers. With reference to this paper, Mr. Lawrence Addicks has communicated to the secretary of the Institute some interesting critical notes which have just been published. Mr. Addicks starts with the statement that the excellent series of photomicrographs given in the paper referred to (pages 89 and 93 of our March issue) shows clearly the relation between micro-structure and cuprous oxide in refined copper. The authors suggest that this method of examination might be substituted for the "button" method now used to control the refining of copper and that, with the aid of a planimeter, it might displace the present tedious methods of analysis for oxygen-content.

Mr. Addicks points out, however, that the photographs of the button-fractures (page 91 of our March issue) show that the fracture at various stages is quite as characteristic as the micro-structure—which is to be expected, as one depends upon the other—and to a practiced eye enables the process to be followed as closely as would be possible with micro-samples, except toward the very end of the poling operation. But at that stage we have the surface or set of full-size castings to guide us. He emphasizes the fact that it is not a certain percentage of cuprous oxide in the casting which is sought, but such a quantity as will give a mechanically perfect product. This quantity varies through a very small range depending upon the size and shape of the casting. The pitch is easily brought close to the proper point with the guidance of the buttons, and any final adjustment made by judging the surface of a larger casting. The pitch is easily brought close to the proper point with the guidance of the buttons, and any final adjustment made by judging the surface of a larger casting. Personally, Mr. Addicks says he cannot see that the use of micro-samples would give any better guidance, and with anode copper in which are varying quantities of numerous impurities, the micro-structure would not be so readily interpreted.

As to the tediousness of the analysis for oxygen, Mr. Addicks says the simplest way to avoid this is to omit it altogether. The buyer of refined copper wants two things—satisfactory electrical conductivity and a mechanically perfect casting. The conductivity is specified because so much copper finds its way into electrical uses. The purchaser relies on the conductivity requirement to give him a copper so pure that it has satisfactory physical properties, and any analysis of refined copper is rarely necessary. Small variations in pitch are noticeable in the set of the castings long before the conductivity is appreciably affected. In regard to the development of the structure of samples of copper, he has had good results by treating the specimens for about 30 seconds in boiling aqua regia diluted with an equal volume of water. Under this condition comparatively little preliminary polishing is required.

American Institute of Electrical Engineers.—The report of the Board of Directors at the recent annual meeting shows that the Institute is in a flourishing condition. With the present month the membership has reached the remarkable figure of 3,000, while it promises to run up very soon to 3,300. The rate of increase in membership is now between 800 and 900 new members per year, which is a very high figure. The building fund for the Institute's share in the Union Engineering Building project has already passed the mark of $60,000, due to the splendid response of members to the request for contributions. The new president is Mr. John W. Lieb, who has been connected with Mr. Edison in his first electrical undertakings; he is at present third vice-president and associate general manager of the New York Edison Co. Like the American Electrochemical Society, the Institute will take a prominent part in the International Electrical Congress in St. Louis in September.

STUDIES ON THE SIEMENS PROCESS FOR WINNING COPPER.

BY M. DEK. THOMPSON, JR.

The following is an abstract of an investigation that was performed in the electrochemical laboratory of the Polytechnikum at Zürich under the direction of Prof. R. Lorenz, during the summer term of 1902 and the winter term of 1902-03. The author wishes to express here his appreciation of the assistance and encouragement given by Prof. Lorenz while this work was carried on.

The two most noteworthy attempts to win copper directly from the ores are the Hoepfner process and the Siemens process. Published investigations on the former of these have appeared,[1] but up to the present time no published account of tests relating to the Siemens process is to be found.

The Siemens process may be briefly described as follows. The ore is ground so as to pass through a fine sieve and then roasted. It is stated by the Siemens firm that roasting is not necessary with all ores. This is then treated at a temperature of 90°, with a solution of ferric sulphate, to which a little sulphuric acid is added. On dissolving the copper, the iron is reduced to ferrous sulphate. This solution of copper sulphate and ferrous sulphate is then led into the cathode compartment of a cell whose anode is separated from its cathode by a piece of linen. Here a part of the copper is deposited on the cathode, and the solution circulates through the diaphragm to the anode. This consists of carbon rods. At the anode the ferrous sulphate is oxidized to ferric sulphate which is used again to dissolve up fresh portions of copper.

THE DISSOLVING ACTION OF FERRIC SULPHATE.

The first part of this investigation consisted in experiments to determine how well ferric sulphate dissolves various copper compounds which were likely to occur in a roasted ore. The experiments were carried out in 100 cc. bottles, which were rotated in a thermostat at 80° C. The thermostat was constant to about a degree.

The ferric sulphate used in these experiments was obtained from Kahlbaum. Complications were introduced by the fact that the sulphate contained free acid. For instance, a solution was made up by dissolving the white powder in water, without adding acid. Analysis showed, however, that instead of 14.460 grams SO_4, corresponding to 5.618 grams iron in 100 cc. of the solution, there were really 20.120 grams SO_4, which corresponds to 5.78 grams free sulphuric acid.

EXPERIMENTS WITH CUPRIC OXIDE.

The first experiments made were with copper oxide. This was obtained from Bender & Hobein. The percentage amount of copper contained was 78.1, as compared with the theoretical 79.9 per cent.

[1] Zeitschrift f. Elektrochemie, 1895, 25; Zeitschrift f. anorg. Chem., 30, 18.

The only possible way ferric sulphate can dissolve copper oxide is by a more or less complete exchange of SO_4 for the oxygen combined with the copper. If in the exchange ferric sulphate loses all its SO_4, the equation would be the following:

(1) $3CuO + Fe_2(SO_4)_3 = 3CuSO_4 + Fe_2O_3$ or

$3CuO + Fe_2(SO_4)_3 + 3H_2O = 3CuSO_4 + 3Fe(OH)_3$.

On the other hand, partial exchange may take place, represented by the following equations:

(2) $2CuO + Fe_2(SO_4)_3 = 2CuSO_4 + Fe_2O_3SO_4$

(3) $CuO + Fe_2(SO_4)_3 = CuSO_4 + Fe_2O(SO_4)_2$

The same equations are obtained by supposing that ferric sulphate reacts with ferric oxide formed in equation (1):

$6CuO + 2Fe_2(SO_4)_3 = 6CuSO_4 + Fe_2O_3$

$2Fe_2O_3 + Fe_2(SO_4)_3 = 3Fe_2O_3SO_4$, giving by addition:

$2CuO + Fe_2(SO_4)_3 = 2CuSO_4 + Fe_2O_3SO_4$

and $3CuO + Fe_2(SO_4)_3 = 3CuSO_4 + Fe_2O_3$

$Fe_2O_3 + 2Fe_2(SO_4)_3 = 3Fe_2O(SO_4)_2$, giving by addition:

$CuO + Fe_2(SO_4)_3 = CuSO_4 + Fe_2O(SO_4)_2$

If basic salts are formed this way, the total amount of SO_4 would never be available for dissolving the copper.

The first experiments given in table 1 were made with 50 cc. of a solution of ferric sulphate containing enough free acid to dissolve 1.862 grams copper, and 2.809 grams iron (except No. 1).

TABLE 1.—TEMPERATURE 80°.

No.	Grm Fe in the Solution.	Grm Cu Dissolved.	Grm Fe Reduced	Duration of Experiment.	Molecular Ratio $CuO + Fe_2(SO_4)_3$.	Grm Cu in the CuO Used.	Residue
1	2.833	1.509	0.032	15 min.	1:1	1.601	....
2	2.809	3.180	0.054	3 hrs.	2:1	3.218	small
3	2.809	..	....	4½ hrs.	4:1	6.436	....
4	2.809	....	....	4½ hrs.	8:1	12 8 8	....

In the first experiment all the copper was presumably dissolved by the acid, since no iron was precipitated. In No. 2 the amount of CuO shaken with the ferric sulphate was increased. After rotating in the thermostate for three hours, a very small quantity of sediment remains undissolved, and the solution was a deep brown. As is seen from the table the amount of copper dissolved was greater than the amount of acid could account for, and still no appreciable amount of iron was precipitated. On acidifying the solution it turned blue. This was the only case where ferric sulphate dissolved copper without itself being precipitated.

In experiment No. 3, where still more CuO was shaken with the same amount of solution, a voluminous brown precipitate containing crystals of copper sulphate was found in the bottle after allowing it to cool. In experiment 4 it was so thick that it could not be poured out of the bottle. The next experiments were therefore made with a solution of ferric sulphate just one-tenth as strong. Table 2 gives the results.

TABLE 2.—TEMPERATURE 80°.

No.	Ratio of $CuO + Fe_2(SO_4)_3$	Gram CuO Used.	Grm Cu Contained by CuO	Grm Fe in 50cc of Solution.	Duration of Exp.	Grm Cu Dissolved.	Residue.	Gram Fe in 50 cc of Solution	Total Amt. Fe in 50 cc.
1	1:1	0.2037	0.1592	0.2809	4 hrs.	0.1592	.....	.....	0.2809
2	3:1	0.6111	0.4776	0.2809	4 "	0.4692	red	0.0073	0.0688
3	5:1	1.018	0.7960	0.2809	4 "	0.6435	reddish brown	.....	.....
4	10:1	2.061	1.609	0.2833	5 hrs. 40 min.	0.3579	brown	.....	0.0058
5	10:1	2.061	1.609	0 2833	2 hrs.	0.6149	brown	.....	.....
6	10:1	2.037	1.592	0.2809	1 "	0.6399	reddish brown	.....	.....
7	10:1	2.037	1.592	0.2809	2¼ hrs.	0.5146	yellowish brown	.....	.....
8	10:1	2.037	1.592	0 2809	5 hrs. 35 min.	0.3643	brown	.....	.....
9	2:1	0.4074	0.3180	0.2809	" "	0.3136	red	0.108	0.1744
10	....	0.2754	0.2205	0.2809	50 min.	0.2205	.....	.....	0.2809
11	....	0.8736	0.6995	0.2809	50 "	0.5099	yellowish brown	.....	.....
12	....	0.8736	0.6995	0.2809	2 "	0.5727	" "	0.108	0.100
13	....	0.8736	0.6995	0.2809	4 hrs.	0.5788	" "	0.0974	0.0933
14	....	0.9590	0.7663	0.2809	5 "	0.7066	" "	.....	.....

Experiment 1. Here there was less than 0.1862 gram copper present. This was dissolved by the acid and the iron remained unchanged.

Experiment 2. Here 0.4692 gram copper was dissolved and 0.2121 gram iron precipitated, but only 0.2830 gram copper was brought into solution by the ferric sulphate. According to equation (1) this amount of iron should have dissolved 0.3613 gram copper, according to equation (2) 0.2404 gram, and according to equation (3) 0.1202 gram. The amount actually dissolved, assuming that 0.1862 gram was dissolved by the free acid, therefore lies between (1) and (2). The small amount of iron in the ferrous state formed undoubtedly comes from impurities in the copper oxide.

In experiment (3), by a similar computation, it was found that the amount of copper dissolved lies between what is required by equations (1) and (2).

Experiment 4. Here both the amount of copper present and the duration of the experiment was increased. In spite of this the amount of copper dissolved was less than in experiment (3). It seemed probable that this was caused by the copper oxide, which still remained unchanged, acting on the copper sulphate to form some basic salt. This supposition was tested in experiments (5) to (8) in which the amount of copper oxide was kept constant, but the duration of the experiment varied. The table shows that with increasing time the amount of copper in solution diminished. All the iron was completely precipitated during the first hour, as shown in the table. The above explanation was therefore verified. The precipitation of copper from a solution of the sulphate by means of copper-oxide, was further tested by shaking 50 cc. of a solution containing 0.6 gram copper, with as much copper oxide as 0.6 gram Cu is contained in. After seven hours' rotation the solution contained 0.4172 gram copper, and a white precipitate had formed.

It was of interest to analyze one of the precipitates that formed in the experiments. Their appearance did not vary much from one experiment to the next. That obtained in experiment 4 gave the following result: 43.98 per cent Cu, 11.13 per cent Fe, 20.67 per cent SO_4.

Analysis in experiments 7 and 8 showed that the amount of SO_4 corresponded with the amount of copper, and therefore no free acid was present. Experiment 10 confirms the assumption that if exactly as much copper is present as the free acid will dissolve, the ferric sulphate remains unchanged.

Experiments 11 to 15 were made to see if it were possible to effect a complete exchange of SO_4 between copper and iron, but it did not prove possible.

In the actual process of extracting the ore ferrous sulphate will always be formed. Before leaving the copper oxide it was desirable therefore to see if this reacted in any way on ferrous sulphate. The following reaction might take place:

$$CuO + FeSO_4 = FeO + CuSO_4.$$

A neutral solution of ferrous sulphate which contained 0.2809 gram iron in 50 cc was shaken with an equivalent amount of copper oxide. 0.1942 gram of copper was dissolved and 0.1906 gram of iron precipitated. The amount of copper is about 10 per cent too little for the above equation. It is probable that this is due to the action of the copper oxide on copper sulphate, just as in the previous cases.

A necessary conclusion from these experiments is that if a large excess of copper oxide is present, all iron and copper will be precipitated. A ferrous sulphate solution was shaken four hours with fifty times the equivalent amount of copper oxide. At the end of this time it was found that all copper and iron had been thrown out.

1. The investigation of the action of ferric sulphate on cupric oxide leads to the conclusion that the reaction between the two is probably represented by the equation:

$$3CuO + Fe_2(SO_4)_3 = 3CuSO_4 + Fe_2O_3,$$

when the two are present in equivalent quantities. When this is not the case, basic salts are formed to a considerable extent.

2. Copper sulphate is precipitated by copper oxide. When the amount of oxide is equivalent to the amount of copper in the solution, the precipitation is only partial; in the presence of a large excess of copper oxide, it is complete.

3. A metathesis takes place between copper oxide and ferrous sulphate, analogous to that with ferric sulphate. Ferrous oxide and copper sulphate are the resulting products.

4. It was shown that under certain conditions all iron and copper salts could be thrown out of solution.

Experiments with Cuprous Oxide.

The next experiments were on cuprous oxide. The sample used was obtained from Bender & Hobein. The percentage amount of copper contained was 83.2 per cent, instead of 88.8 per cent. There was a trace of manganese, which was precipitated on the anode during electrolysis.

The first experiment is tabulated below:

Table 3.

Wt. Cu_2O	Wt. Cu in Cu_2O	Ratio of Cu_2O + $Fe_2(SO_4)_3$	Gram $Fe^{...}$ in Solution.	Dura-ti'n of Exp.	Temp.	Gram Cu Dissolved.	Grm. Fe Reduced.
1.934	1.609	1:2	2.833	17 min.	80°	all	1.34

The amount of iron that should have been reduced by 1.609 grams copper, assuming the equation,

$$Cu_2O + Fe_2(SO_4)_3 + H_2SO_4 = 2CuSO_4 + 2FeSO_4 + H_2O,$$

is 1.41 grams, which is more than that found. This difference of 5 per cent is easily explained by the probability of the presence of some cupric oxide which would be dissolved by the acid alone. In an experiment using twice as much cuprous oxide the copper sulphate partly crystallized out on cooling, so that the following experiments were made with a solution of ferric sulphate ten times as dilute. The results are given in table 4.

Table 4.

No.	Ratio of Cu_2O + $Fe_2(SO_4)_3$	Wt. Cu_2O	Wt. Cu in Cu_2O	Wt. $Fe^{...}$ in 50cc of Solution.	Grm Cu Extracted.	Residue.	Grm $Fe^{...}$ in 50cc of Solution after Extraction.	Total Amount Iron in 50cc	Duration.
1	1:2	0.1913	0.1592	0.2809	all	.....	0.1240	0.2809	4 hrs.
2	1½:2	0.2869	0.2388	0.2809	all	.....	0.1717	0.2809	4 "
3	3:2	0.5739	0.4776	0.2809	0.4302	brown	0.2352	0.2347	4 "
4	6:2	1.1478	0.9552	0.2809	0.4443	red	0.2219	0.2210	4 "
5	10:2	1.9354	1.609	0.2833	0.4723	.....	0.272	0.205	5 hrs. 40 min.
6	1:1	0.3661	0.3206	0.2822	all	.....	0.2550	0.2822	several min.
7	1:1	0.7322	0.6412	0.2822	all	.....	0.2583	0.2822	" "
8	1:1	0.732	0.6412	0.2822	all	.....	0.2709	0.2822	1 hour
9	1:1	0.805	0.705	0.2822	0.2780	.....	0.2368	0.2822	several min.

In the first experiment there was an excess of ferric sulphate present. All the copper present was dissolved, and an amount of iron reduced which was less than that computed by the equation above. The same is the case in experiment 2.

In experiment 3 the amount of iron required by the equation was not present. Therefore, after reducing all the iron the reaction was carried a step farther, and some iron was precipitated. With cuprous oxide there was never as much as one-half of the iron precipitated. It was also found that the iron still in solution was in the ferrous form. This fact, and the presence of ferrous iron in the precipitate makes the following process probable:

1. Copper is first oxidized: $Cu_2O + Fe_2(SO_4)_3 = CuO + CuSO_4 + 2FeSO_4$.
2. $CuO + FeSO_4 = CuSO_4 + FeO$.

Equation 2 has already been tested, and it was found later that cuprous oxide and ferrosulphate scarcely have any effect on each other.

In experiment 5 the amount of cuprous oxide was increased with the result that more iron was precipitated. In order to test the equation,

$Cu_2O + Fe_2(SO_4)_3 + H_2SO_4 = 2CuSO_4 + 2FeSO_4 + H_2O$,

more exactly, the next experiments were made with some oxide obtained from Kahlbaum. It contained 87.6 per cent Cu instead of the theoretical amount 88.8 per cent. In experiments 6, 7, 8 and 9 cuprous oxide and ferric sulphate were present in equivalent amounts. The solution took place so rapidly that all the copper was dissolved before the bottle could be corked and placed in the thermostat. These experiments show that when air is more completely excluded, results can be obtained that agree better with values calculated from the formula. In No. 6, for example, in which more than 50 cc. of air was present, the result agrees with the formula only to 10 per cent, while in No. 9 it agrees to 3 per cent. In the last case the bottle contained only 15-20 cc. of air, and the solution was effected by shaking in the hand for a few minutes. The amount of oxygen present in this amount of air could oxidize 0.04 gram of iron, which fully accounts for this discrepancy. The experiments show then that the above equation is correct.

It still remained to investigate the effect of copper sulphate and ferro sulphate on cuprous oxide. Fifty cc. of a solution containing 0.600 gram Cu were shaken eight hours with 4 grams of Kahlbaum's cuprous oxide. The amount of copper in solution was then found to be 0.557 gram, or 0.04 gram less than at the start. On account of the small difference it seems probable that the decrease is due to impurities in the copper oxide, so that these substances have no effect on each other, or at least only a very slight one. A similar result was obtained with ferro sulphate. 100 cc. of a solution containing 0.7466 gram iron were shaken three hours at 80°, with about 2 grams of cuprous oxide; 0.7086 gram ferrous iron was then found in the solution. The small amount of iron thrown out is apparently due to impurities also.

The results of the experiments with cuprous oxide are:

1. Cuprous oxide reduces ferric sulphate to ferrous sulphate according to the equation,

$Cu_2O + Fe_2(SO_4)_3 + H_2SO_4 = 2CuSO_4 + 2FeSO_4 + H_2O$,

and both ferrous and ferric iron is precipitated.

2. Cupric sulphate does not act on cuprous oxide.
3. Ferrous sulphate does not act on cuprous oxide.

EXPERIMENTS WITH CUPROUS SULPHIDE.

The equation which is given for the action of ferric sulphate on cuprous sulphite is the following:*

$Cu_2S + 2Fe_2(SO_4)_3 = 2CuSO_4 + 4FeSO_4 + S$.

The object of this part of the investigation was to test this equation.

The sulphide first used was Merk's, which came in the form of square rods. This was ground up in a mortar and shaken through a No. 80 sieve (80 meshes per inch). Analysis gave the following results:

	Found.	Theoretical Amount.
Cu	78.4%	79.86
S	21.5%	20.14

For the first experiments 50 cc. of the stronger solution of ferric sulphate were used. The following table gives the result:

Ratio of Cu_2S+ $Fe_2(SO_4)_3$	Grm Cu_2S	Grm Cu in Cu_2S	Grm $Fe^{...}$ in the Solution	Duration of Exp.	Grm Cu Dissolved.	Grm Fe Reduced.	Computed Amt. Fe Reduced
1:2	2.052	1.609	2.833	¼ hr.	0.9600	1.675	1.69

A comparison of the amount of iron reduced, and the computed amount shows a fair agreement. Some cuprous sulphide was then prepared by precipitation of cupric sulphide from a copper sulphate solution and reduction in a Rose crucible in a stream of hydrogen. Table 5 gives the results obtained with this sulphide.

TABLE 5.

No.	Grm Cu_2S Used.	Grm Cu Contained in Cu_2S	Grm $Fe^{...}$ in Solution.	Duration of Exp.	Grm Cu Dissolved.	Grm Fe Reduced.	Computed Amt of Fe Reduced	Vol. of solution.
1	0.876	0.699	0.3822	2 hrs. 5 min.	0.2280	0.283	0.402	50
2	1.752	1.399	0.2822	" "	0.2208	0.286	0.389	50
3	0.946	0.756	0.5644	¼ hr.	0.3168	0.3546	0.5678	100
4	0.946	0.756	0.5644	6 hrs.	0.3288	0.3645	0.5700	100

In experiments 1 and 2 the amount of copper dissolved should have reduced more iron than was present. This was caused by the oxygen of the air, as the two following experiments prove. Here the bottles were nearly completely filled, and the water used for making the solution of ferric sulphate was boiled. Under these circumstances the amounts of iron found agrees with those computed to about 1 per cent. The amounts found were still always smaller than computed, which may be attributed to the small amount of oxygen still present.

For the sake of completeness the experiments of table 6 were made to see if sulphuric acid has any effect on cuprous sulphide. Best conducting sulphuric acid was used.

TABLE 6.

Gram Cu_2S.	cc. H_2SO_4.	Duration of Experiment.	Temp.	Gram Cu Dissolved in 50 cc.
1.0	100	5¼ hrs.	80°	0.0085
0.5	50	5¼ hrs.	80°	0.0468

These results show that, in the presence of oxygen, cuprous sulphide is dissolved to a slight amount.

The following experiments were made to find whether any reaction takes place between cuprous sulphide and ferrous sulphate or copper sulphate: 100 cc. of a copper sulphate solution containing 1.194 grams copper were shaken six hours at 80°, with 5 grams of Merk's cuprous sulphide; 1.186 grams copper were then found in the solution, which is an inconsiderable decrease.

The ferrous sulphate used in the second experiment was made up with boiled water and contained 0.2849 g. iron in 50 cc. 100 cc. were shaken for 2¾ hours at 80°, with 0.923 g. cuprous sulphide. The solution was tested with hydrogen sulphide and found to contain no copper, and the analysis for iron gave a result agreeing to 0.3 per cent, with the original amount.

The results of these experiments are:

1. The verification of the equation,

$Cu_2S + 2Fe_2(SO_4)_3 = 2CuSO_4 + 4FeSO_4 + S$.

2. The $FeSO_4$ formed has no effect on cuprous sulphide.
3. The $CuSO_4$ formed has no effect on cuprous sulphide.
4. Sulphuric acid dissolves cuprous sulphide in the presence of oxygen very slowly. The presence or absence of sul-

* See Borchers "Elektrometallurgie."

phuric acid in the solvent is therefore of very little consequence as far as Cu_2S is concerned.

Experiments with CuS.

The following seems to be the simplest reaction that can take place between copper sulphide and ferric sulphate:

$$CuS + Fe_2(SO_4)_3 = CuSO_4 + 2FeSO_4 + S.$$

The two experiments of table 7 were made to test this equation. The copper sulphide used was prepared from copper sulphate by precipitation with hydrogen sulphide.

Table 7.

No.	Grm CuS Used.	Grm Cu in CuS	Grm Fe''' in Solution.	Temp.	Duration of Exp.	Grm Cu Dissolved.	Grm Fe Reduced.	Computed Amt. of Iron Reduced.
1	0.4824	0.3390	0.5644	80°	6 hrs.	0.2495	0.3520	0.4393
2	1.053	0.6995	0.3822	80°	2 hrs.	0.2377	0.2798	0.4186

In experiment 1, where the bottle was nearly full, and therefore not much air was present, the amount of iron reduced was less than that computed by the above equation, and in experiment 2, where only 50 cc. of the solution were used the discrepancy was still greater. In one case the residue was treated with carbon bisulphide and filtered. After evaporating a small amount of sulphur was left behind, proving that sulphur is set free in the reaction, and that so far at least the above equation is correct. If this equation is correct, however, the excess of copper must have been dissolved some other way. The following experiment showed that this excess was due to the oxygen of the air oxidizing copper sulphide to sulphate. In a 100 cc. bottle containing 50 cc. water and 1 gram copper sulphide the air was replaced by oxygen. After 5½ hours at 80° 0.171 gram copper was dissolved. Table 8 gives the results of experiments with sulphuric acid, copper sulphate and ferro sulphate:

Table 8.

Solvent.	Volume.	Wt. CuS Used.	Duration of Experiment.	Gram Cu Dissolved in 100 cc. of Solvent.
H_2SO_4 Sp gr. 1.2....	100 cc.	1.00	3 hrs.	0.0688
1.15% $CuSO_4$ Sol....	100 cc.	1.00	6 hrs.	0.079
0.55% $FeSO_4$ Sol....	100 cc.	1.00	6 hrs.	0.071

In all cases about the same amount of copper was dissolved, which shows that it was due to the oxygen present. Copper sulphate is therefore not dissolved by these reagents.

Experiments with Iron Oxide

Many copper ores contain iron, which is changed over to the oxide on roasting. It was therefore of interest to see if iron oxide is dissolved by ferric sulphate. 2.25 grams Fe_2O_3 were shaken for about an hour at 80°, with 50 cc. of a solution that contained 2.83 gr. iron. There was also some free acid present that came from the ferric sulphate. The solution was then analyzed and 2.79 gram iron found. A similar experiment was made with ferric hydrate freshly precipitate from 10 cc. of a 5 per cent solution. It was washed till the wash water gave no reaction for sulphuric acid, and was then shaken with 50 cc. of a solution containing 0.282 gram iron. The residue was then filtered off and well washed. It was found in dissolving some of this residue in hydrochloric acid and testing with barium chloride that some sulphate was present, indicating a basic salt. The reaction that takes place between Fe_2O_3 and $Fe_2(SO_4)_3$ is therefore to precipitate iron from the solution.

The Precipitated Sulphate.

Since sulphur is liberated from sulphide ores in the process of leaching, it was necessary to see whether this can be oxidized to sulphuric acid by ferric sulphate. In order to obtain sulphur in the same state as that liberated by leaching, some cuprous sulphide was warmed four times with fresh portions of 6 per cent ferric sulphate solution. The residue, which consisted mostly of sulphur, was washed, dried and then shaken with a new sample of 50 cc. of the same solution for about five hours at 80°. This solution contained 213.1 grams SO_4 per l; after shaking with free sulphur 216.0 grams SO_4. The ferric sulphate therefore oxidized a very inconsiderable amount of sulphur; and therefore this reaction can have no significance for this process.

Experiments with SO_2

Some experiments were made on the relative efficiency of reducing ferric sulphate with SO_2, and with this gas largely diluted, as in the case of furnace gasses. Leading the pure gas (taken from a bomb) through ferric sulphate at 80°, the amount of iron reduced was 8.9 per cent of that which would have been reduced, if all the SO_2 had been used up, as was found by titrating the iron with permanganate after driving out the SO_2 by CO_2 gas. On diluting the SO_2 with air so that it amounted to about 10 per cent of the total amount, 5 per cent iron was reduced. The furnace gases might, therefore, be used to reduce the small amount of ferric sulphate still remaining after having treated the ore, as the solution from which the copper is to be precipitated should contain no ferric salt.

Experiments with a Copper Ore.

Analysis of the ore gave the following results:

Cu	29.99%
Fe	27.89%
S	33.32%
SiO_2	9.75%
	100.95

Leaving out the silicon dioxide the relative amounts of copper, iron and sulphur are given in table 9, and the composition of copper pyrites is also tabulated for the sake of comparison.

Table 9.

	Contained in $CuFeS_2$.	Contained in the Ore.
Cu	34.4%	32.9%
Fe	30.6%	30.6%
S	34.9%	36.5%

The composition of the ore corresponds therefore closely to that of copper pyrites. The first experiments showed that the ore was scarcely attacked by a 5 per cent solution of ferric sulphate. The powdered ore was therefore shaken through a No. 80 sieve and roasted several hours in an enamel plate over a Bunsen burner. It was then found to contain 28.8 per cent Cu and 18.9 per cent S. From the previously determined ratio of copper to iron the amount of iron present in the roasted sample was computed to be 27.0 per cent.

Some of the iron and copper has been oxidized, so that 7.6 per cent of the ore was copper sulphate, and 2.3 per cent iron sulphate. In table 10 the process of solution can be followed: 5.351 grams ore and 50 cc. of a ferric sulphate solution containing 2.833 grams iron were used. The ore contained 1.54 grams copper.

Table 10.—Temperature 83° C.

	After ¼ hr.	After 1 hr.	After 2 hrs. 10 min.	After 6 hrs.
Iron Cu dissolved.....	0.9045	1.047	1.118	1.163
Per cent Cu dissolved.	58.7	68.3	72.5	75.3
Grams Fe dissolved...	0.237	0.407	0.470	0.440
Per cent Fe dissolved.	16.5	27.2	31.4	29.4
Grams Fe''' present..	2.080	2.001	1.908	1.751

These results show that not much more than 75 per cent of the copper could be dissolved from this sample. The curves in Fig. 1 represent this table graphically. Curve 1 represents the ferric sulphate present. The straight line below curve 1 represents the original amount of copper in the ore. Curves 2 and 3 represent the copper and iron dissolved, respectively.

A second set of experiments was made with a smaller amount of ore, and the same volume of solvent. In this case a little more copper was dissolved, as the following table and curves show.

2.398 grams ore was used, containing 0.6899 gram Cu and 0.6483 gram Fe. The solvent was the same. Table 11 and Fig. 2 give the results.

TABLE II.—TEMPERATURE 80°.

	After ¼ hr.	After 55 min.	After 1 hr. 55 min.	After 6 hrs. 15 min.
Iron Cu dissolved ...	0.4115	0.4700	0.4797	0.5415
Per cent Cu dissolved.	59.6	68.0	69.5	78.5
Gram Fe dissolved..	0.107	0.195	0.191	0.239
Per cent Fe dissolved.	16.5	30.0	29.8	36.9
Gram Fe$^{\cdots}$ present.	2.312	2.033	2.024	1.819

In Fig. 2, curve 1 represents the ferric sulphate present, curve 2 the copper dissolved, curve 3 the iron dissolved, while the straight line represents the copper in the ore at the start.

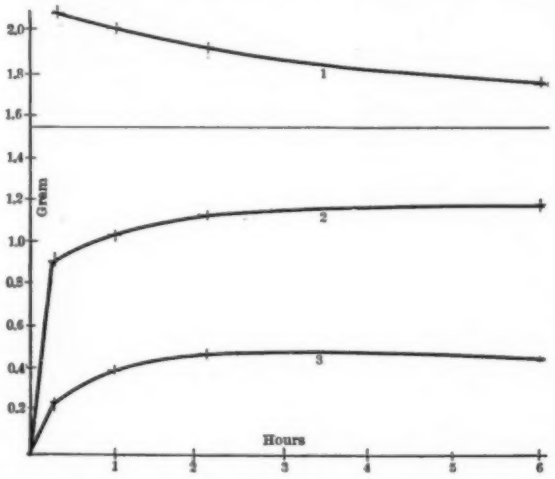


FIG. 1.—CURVE 1, FERRIC SULPHATE PRESENT; CURVES 2 AND 3, COPPER AND IRON DISSOLVED RESPECTIVELY.

These results confirm those previously obtained. Experiments were then performed to see if more of the copper could be dissolved out of the ore already treated by using fresh portions of solvent. Three bottles were filled with 100 cc. of a 6 per cent solution of ferric sulphate and 12 grams of ore, and shaken for two hours at 80°. The amounts of copper dissolved were 65.6 per cent, 66.2 per cent, and 66.2 per cent. The solution was siphoned off and 100 cc. fresh solvent added

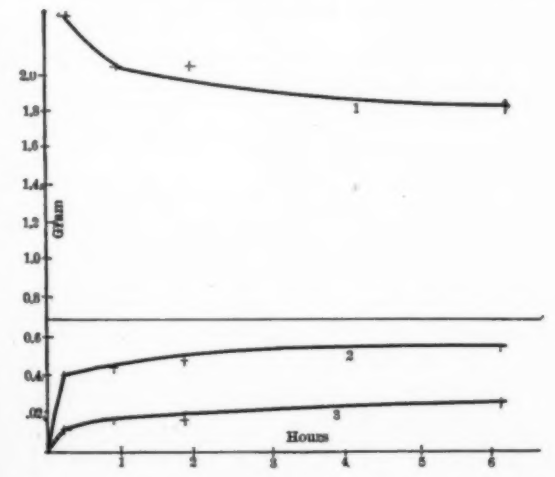


FIG. 2.—CURVE 1, FERRIC SULPHATE PRESENT; CURVE 2, COPPER DISSOLVED; CURVE 3, IRON DISSOLVED.

to each bottle. They were then shaken for different lengths of time, as shown in table 12.

	Bottle No. 1. Shaken 1 hr.	Bottle No. 2. Shaken 2½ hrs.	Bottle No. 3. Shaken 5 hrs.
Grm. Cu dissolved	0.193	0.256	0.350
Per cent Cu dissolved in this and previous treatments	71.2	73.6	76.3

The bottles were again filled and No. 1 was shaken one hour, No. 2 two and one-half hours, and No. 3 five hours. This increased the amount dissolved in No. 1 to 73.5 per cent, in No. 2 to 76.4 per cent, and in No. 3 to 79.6 per cent. The last bottle was shaken again with fresh solvent for five hours, which increased the amount of dissolved copper to 81 per cent. The residues from these experiments were collected and roasted again. It was found that by this means 91.8 per cent of the copper still remaining in the ore were dissolved by the solvent in an hour. For this reason the ore used in the above experiments was roasted again. The copper then amounted to 30.2 per cent. Six grams of this ore, when shaken one hour at 80°, with 100 cc. of approximately 6 per cent ferric sulphate solution. This dissolved 1.702 grams, or 93.9 per cent of the copper present. In five hours, under similar conditions, 98.9 per cent copper was dissolved.

These experiments show that copper can be dissolved by ferric sulphate pretty completely as soon as the ore is sufficiently roasted. The question is what are the compounds formed by roasting, which are so much more soluble than the unroasted ore. It seems that this is due to the formation of copper oxide. If this is the case, just as much copper would be dissolved by sulphuric acid. To test this conclusion 6 grams of the ore used in the last experiments were shaken with 100 cc. of sulphuric acid, sp. gr. 1.2, for five hours. This dissolved 1.71 grams, or 97.7 per cent. It was found, however, in the study of cuprous sulphide, that this is easily extracted by ferric sulphate, and this is exactly what was under investigation in these experiments, combined with the sulphide of iron. It must be this double combination therefore that prevents the copper from being extracted by the ferric sulphate.

The amount of sulphate was slightly increased by the second roasting. 8.9 per cent of the ore was copper sulphate, which corresponds to 1.8 per cent sulphur. As the total amount of sulphur was found to be 8.2 per cent, there must have been a considerable amount present in the form of sulphide. It could not be determined, however, whether it was combined with the iron or the copper. It was not present in iron sulphate, as only 0.4 per cent of the iron was in the form of sulphate.

The results of this part of the investigation may be summed up as follows:

1. Copper pyrites is not appreciably attacked by ferric sulphate.

2. Roasting so changes the ore that nearly all the copper can be extracted either by sulphuric acid or by ferric sulphate. This makes it seem probable that the roasting changes the copper largely over to the oxide.

THE ELECTROLYSIS.

The electrolysis comprises two reactions, the reduction of the copper on the cathode and the oxidation of iron at the anode. These were investigated separately. The vessel used was a copper voltameter jar divided into three narrow compartments by two clay diaphragms made fast to the glass sides with paraffin. The anode and cathode were in the outer compartment, while the inner one prevented diffusion from one electrode to the other.

In the experiments on the deposition of copper a lead plate was used as anode, and the anode compartment was filled with sulphuric acid. The cathode compartment was filled with a solution of the following composition:

5 per cent ferrous iron.
3.5 per cent copper.
2.5 to 3 per cent sulphuric acid.

The same solution was contained in the middle compartment. During electrolysis the liquid in the cathode compartment was stirred by a current of carbon dioxide.

The object in these experiments was to determine how poor the solution may become in copper without affecting the character of the copper deposited at a given current density. This was to be determined for different current densities. For this purpose the gain in weight of the copper cathode was compared with the copper deposited in a copper voltameter. Table 13 gives the results obtained with a current density of 0.98 ampere per square decimeter.

TABLE 13.

Time in Hours.	Cu Deposited in Voltameter for ½ hr.	Cu Deposited in Cell for ½ hr.	Per Cent Yield.
½	0.3346	0.3301	98.65 per cent.
1	0.3346	0.3328	99.45 "
1½	0.3377	0.3356	99.40 "
2	0.3385	0.3384	100 "
2½	0.3474	0.3469	99.97 "
3	0.3397	0.3399	100 "
3½	0.3321	0.3312	99.75 "
4	0.3340	0.3341	100 "
4½	0.3399	0.3352	.....

After four hours the copper commenced to be spongy. The cathode solution was then analyzed and was found to contain 0.72 per cent copper.

In another similar experiment with the same current density, the copper did not begin to be spongy till the strength of the solution had reached 0.38 per cent copper.

Table 14 gives the results with a current density of 1.8 ampere per square decimeter.

TABLE 14.

Time in Hours.	Cu Deposited in Voltameter in ½ hr.	Cu Deposited in Cell in ½ hr.	Yield.
½	0.6049	0.5934	98.1 per cent.
1	0.6322	0.6157	97.4 "
1½	0.6123	0.6086	99.4 "
2	0.6038	0.6020	99.7 "

At the end of the second hour the copper began to be spongy. The electrolysis was therefore discontinued and the solution in the cathode compartment analyzed. It contained 0.98 per cent Cu.

The current density was then reduced to 0.47 ampere per square decimeter, and the electrolysis continued, with the results given in table 15.

TABLE 15.

Time in Hours.	Cu Deposited in Voltameter in ½ hr.	Cu deposited in Cell in ½ hr.	Yield
½	0.1689	0.1693	100 per cent.
1	0.1589	0.1547	97.35 "
1½	0.1611	0.1595	99.02 "
2	0.1646	0.1622	98.55 "
2½	0.1639	0.1639	97.7 "
3	0.1658	0.1669	100 "

Here the copper came down in good form till the concentration of copper was 0.05 per cent in the solution.

Two other experiments were carried out with current densities of 3.4 amperes and 2.6 amperes per square decimeter, respectively. In the first case the copper was deposited in a spongy form immediately. The same was true in the second case, though not in so marked a manner. Some of the spongy copper was tested for iron, but this impurity was not present.

The results are best shown by Fig. 3 in which the abscissae represent the current density, and the ordinates the concentrations of the copper solution when the copper began to be spongy.

OXIDATION OF THE ANODE.

In these experiments the anodes consisted of round carbon

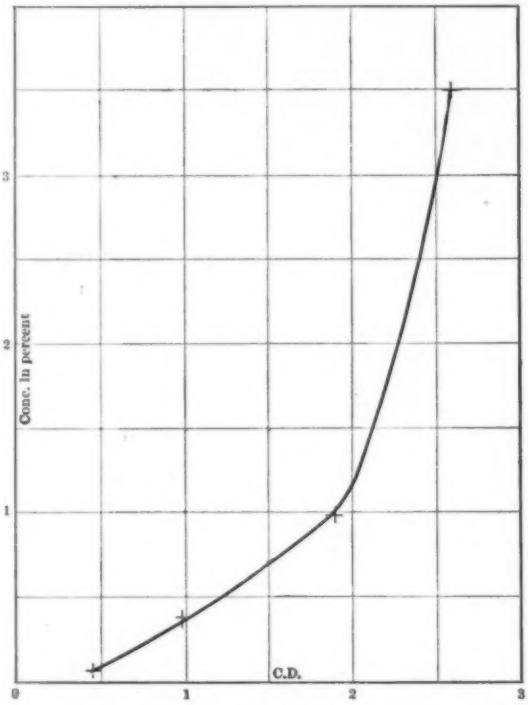


FIG. 3.—CURVE SHOWING THE CONCENTRATION OF THE COPPER SOLUTION FOR WHICH THE COPPER BEGINS TO BE SPONGY AT A CERTAIN CURRENT DENSITY.

rods 2 cm. in diameter and 15 cm. long. Each rod stood in a clay cylinder 12.5 cm. high and 4 cm. in diameter, which were filled with the anode solution. The clay cylinder was placed in a large glass vessel. A cylindrical sheet of copper served as cathode, and copper sulphate as the catnodic solution. Several such cells were connected in series and electrolyzed. From time to time a cell was disconnected and analyzed, while the electrolysis of the rest was continued. The amount of iron oxidized was compared with a copper voltameter connected in series with the cells. The amount of iron oxidized was determined by analyzing the amount of ferrous iron in the anode compartment before and after the electrolysis. The amount of ferrous iron diffused into the cathode compartment was determined and allowed for.

Table 16 gives the results with a current density of 0.48 ampere per square decimeter. The current was 0.36 ampere, and the surface of the anode 0.76 square decimeter.

TABLE 16.

Time.	Gr Fe" in Anode Compartment.	Gr Fe" Diffused from Anode Compartment.	Gr Fe" Oxidized.	Gr Cu in Voltameter.	Gr Fe Representing 10% Yield.	Yield of Oxidiz'd Iron.
						Per Cent
9:15	2.776	.....	.....	.....	.....	....
10:15	1.933	0.0671	0.776	0.419	0.738	105
11:15	1.451	0.1122	1.213	0.7642	1.344	90
12:15*	1.058	0.104	1.614	1.143	2.015	79.5
2:30	0.6676	0.1815	1.926	1.581	2.785	69
3:30	0.2446	0.1724	2.359	1.971	3.47	68
4:30	0.1044	0.189	2.483	2.417	4.25	58.3

* Electrolysis interrupted from 12:15 to 1:30.

In this series, as in others, the yield for the first period amounted to more than 100 per cent. It is possible that the

oxygen absorbed in the carbon may have been the cause for this. These results are shown graphically in Fig. 4.

Table 17 contains the results of the second series, with a current density of 0.26 ampere per square decimeter, while Fig. 5 shows the results graphically.

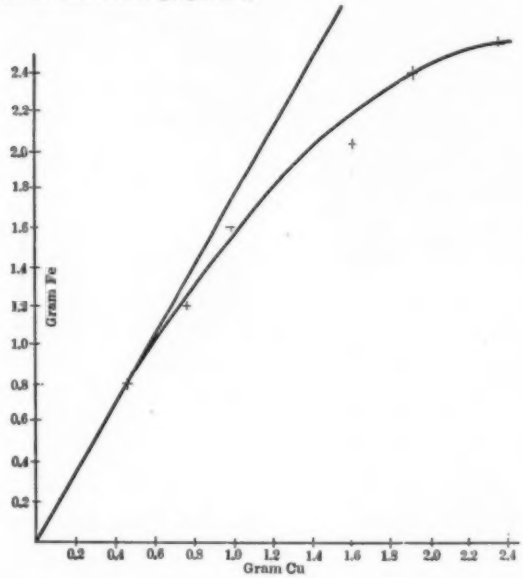


FIG. 4.—THE ABSCISSAS ARE THE GRAMS Cu IN THE COPPER VOLTAMETER, THE ORDINATES IN Gr Fe OXIDIZED. THE STRAIGHT LINE GIVES THE THEORETICAL YIELD ACCORDING TO FARADAY'S LAW. THE BENT CURVE SHOWS THE RESULTS OBTAINED EXPERIMENTALLY, C.D. = 0.47.

TABLE 17.

Time.	Gr $Fe^{..}$ in Anode Compartment.	Gr $Fe^{..}$ diffused from Anode Comp.	Sum of Preceding two Columns.	Gr Iron Oxidized.	Cu Precipitated in Voltameter.	Gr Iron Representing 100 per ct. Yield.	Yield.
							Per Ct.
10:23	2.791	------	------	------	------	------	------
11:30	2.097	0.013	2.110	0.681	0.3641	0.641	106
11:45	1.629	0.055	1.684	1.107	0.5780	1.02	108
2:35	1.219	0.060	1.279	1.512	0.8715	1.54	98
3:40	0.7849	0.0953	0.8502	1.941	1.396;	2.35	84
4:30	0.4317	0.1060	0.5397	2.251	1.6134	2.84	79
5:05	0.2715	0.1249	0.3964	2.393	1.9300	3.40	70

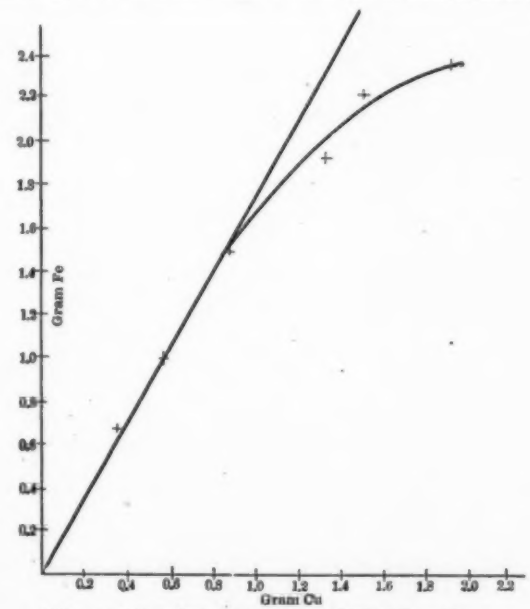


FIG. 5.—THE SAME AS IN FIG. 4, BUT C.D. = 0.26.

A third experiment was carried out with a current density of 0.53 ampere per square decimeter, the results of which are tabulated in table 18. At the end of the first and last periods two cells were taken out together and analyzed, in order to see how well they would agree when electrolyzed for exactly the same length of time. The results for the first period agreed to 6 per cent, and for the last to 4 per cent.

TABLE 18.

Time	Gr $Fe^{..}$ in the Anode Compartment.	Gr $Fe^{..}$ Diffused Away.	Sum of two Preceding Columns.	Gr Fe Oxidized.	Cu Precipitated in Voltameter.	Gr Fe Representing 100 per ct. Yield.	Yield.
							Per Ct.
9:00	2.779	------	------	------	------	------	------
9:40	2.116 / 2.072	0.097 / 0.107	2.213 / 2.179	0.566 / 0.600	0.3273	0.559	101 / 107
10:55	1.456	0.114	1.370	1.409	0.8895	1.52	92.6
11:45	0.786	0.152	0.920	1.650	1.2478	2.14	77.6
1:48	0.207 / 0.108	0.267 / 0.278	0.474 / 0.386	2.305 / 2.39	2.0820	3.56	64.8 / 67.2

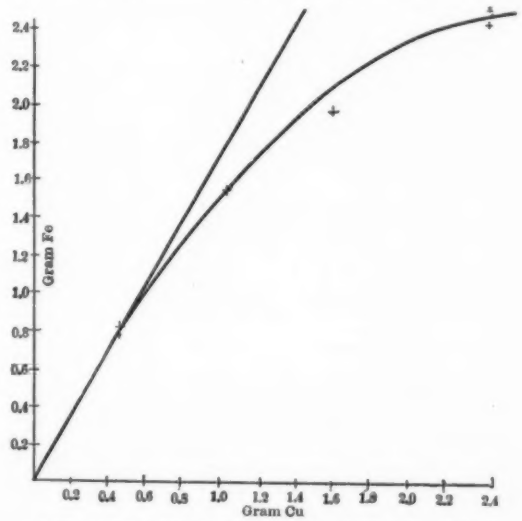


FIG. 6.—THE SAME AS IN FIG. 4, BUT C.D. = 0.53.

As Fig. 6 shows, the deviation from Faraday's law becomes marked at about the same time for this current density as for that of the preceding runs, showing that a variation from 0.3 ampere to 0.5 ampere per square decimeter has no noticeable effect on the current yield. Large deviations from Faraday's law were accompanied by a strong evolution of gas.

In all these experiments the yield for the first period amounted to more than 100 per cent. It seems probable, therefore, that if the disturbing causes were removed, the yield would not fall much below 100 per cent. At the end of the first period the solution contained 2.1 per cent ferrous sulphate. Therefore, the conclusion may be drawn that starting with a solution containing 5 per cent ferrous iron, this may be oxidized with approximately 100 per cent yield, till the solution contains only 2 per cent of ferrous iron, using a current density from 0.3 to 0.5 ampere per square decimeter. When the solution contains between 2 per cent and 1 per cent of oxidizable iron, the yield would still be about 90 per cent. When the concentration is still more diminished, the evolution of gas becomes strong and the yield falls off correspondingly.

AMERICAN INSTITUTE OF MINING ENGINEERS.—The eighty-seventh meeting of the Institute will be held in September in the Lake Superior region, and arrangements will be made for a subsequent visit of members to St. Louis in the last week of September, with a supplementary session at the World's Fair. Headquarters for members of the American Institute of Mining Engineers and the American Society of Mechanical Engineers will be maintained in the Mining Building.

WET METHODS OF EXTRACTING COPPER.

In a recent paper, presented to the *American Institute of Mining Engineers*, considerable information is given by Mr. Charles H. Jones on the methods of leaching and precipitating copper, in use at Rio Tinto in Spain. This paper is of special interest and value, in view of the fact that most of the information heretofore published pertaining to these methods has become obsolete.

The ore treated at Rio Tinto is a massive iron pyrite containing up to 3 per cent of copper, which has been disseminated through the mass by a secondary enrichment. The well-known method adopted for the extraction of the copper consists simply in allowing huge heaps of the mineral to oxidize under the influences of moisture and air, and subsequently washing out the copper sulphate as soon as it is formed, by running water through the heap.

The successful and economical application of this system depends largely on the state in which the copper occurs in the mineral. If it exists as chalcopyrites, $CuFeS_2$, the copper will not oxidize by simple exposure to the air. If the copper is in the form of CuS, the oxidation proceeds very slowly. The best form for solution is Cu_2S or copper glance, which constitutes the bulk of the copper in Rio Tinto pyrite.

The following reactions take place during oxidation. When the mineral is oxposed to free access of air and moisture, some ferrous sulphate is formed as follows:

(1) $FeS_2 + 7O + H_2O = FeSO_4 + H_2SO_4$.

The ferrous sulphate thus formed becomes readily oxidized by the air to ferric sulphate.

(2) $2FeSO_4 + H_2SO_4 + O = Fe_2(SO_4)_3 + H_2O$.

and it is due to the action of this ferric sulphate on the copper sulphides that the copper is rendered soluble, as represented by the following equations:

(3) $Fe_2(SO_4)_3 + Cu_2S = CuSO_4 + 2FeSO_4 + CuS$.

(4) $Fe_2(SO_4)_3 + CuS + 3O + H_2O = CuSO_4 + 2FeSO_4 + H_2SO_4$.

The reaction (3) takes place fairly rapidly and causes half the copper to go into solution within a few months, while the reaction (4) proceeds much more slowly, and requires, under the most favorable conditions, about two years to extract 80 per cent of the remaining half of the copper.

In practice the method adopted to bring about the desired oxidation is as follows: A site is chosen for the formation of the heap where the ground is sufficiently concave and sloping to enable the copper liquor that is formed to collect and run out at the base of the heap. On the ground is first arranged a network of air-flues made of rough stones and having an internal diameter of 12 inches. Vertical chimneys, 50 feet distant from one another, are built in the same manner and connect with the ground flues. Care is taken that the mouth of the ground flues are kept open and not covered by the ore. The mineral, the lump portion of which has been passed through jaw-breakers to be reduced to pieces not larger than from 2 to 3 inches across, is now tipped from side-tip wagons at the highest part of the selected side over and around the stone flues. "Lump" and "fines" are alternately dumped until the height of the mass at the edge is about 30 feet, the upper surface of the mineral being kept level. A heap of this form approximately contains 100,000 tons of ore.

As the mineral is added, the building of the stone chimneys keeps pace, in order to have a clear opening to the top of the heap. The surface of the heap is formed into squares by means of ridges of the mineral, the size of these squares depending on the porosity of the heap. The function of these ridges is to enable the water to be run on locally over the surface of the heap in order to insure that all parts are equally washed, and that the water does not run through the heap in channels. A system of gutters is also arranged so that the water can be run on to all parts of the mass. As the heap is being formed, water is run on, and the copper sulphate existing in the mineral is extracted; also, the water provides the moisture for the oxidation to take place in accordance with the equations given above. The mineral in the heap is then allowed to oxidize, which it does pretty rapidly, as evidenced by the heat produced, the temperature in the chimneys rising to from 170° to 180° F. As the temperature increases, the surface-openings of the chimneys may be closed in order to allow the oxidation to spread through the heap.

The surface gradually shows a brownish coloration due to the dehydration of the buff-colored basic ferric salt that forms on the top of the mass and its gradual heating up may be noted by this drying action. The greatest care must be taken not to allow the heap to fire, for if once started it is very difficult to extinguish it. When the oxidation has proceeded as far as it is safe to allow it, water is run on at the rate of about 50 cu. m. per hour, until the soluble copper is leached out; the heap is then allowed to re-oxidize and the washing is repeated. After about a year has elapsed the surface requires "retilling," and the squares are re-arranged so that the places where the ridges were before are now the middles of the squares; the gutters also are shifted. At the edge of the heap for a distance of some yards the mineral which has become cemented holds a considerable quantity of copper salts and is dug down into terraces in order that this copper may be extracted by washing. When the copper is reduced to 0.3 per cent, the heap is considered washed and the mineral, containing 49.5 per cent of sulphur, is removed and exported as "washed sulphur ore" and utilized for the manufacture of sulphuric acid. Fig. 1 shows the surface of the heaps arranged for washing.

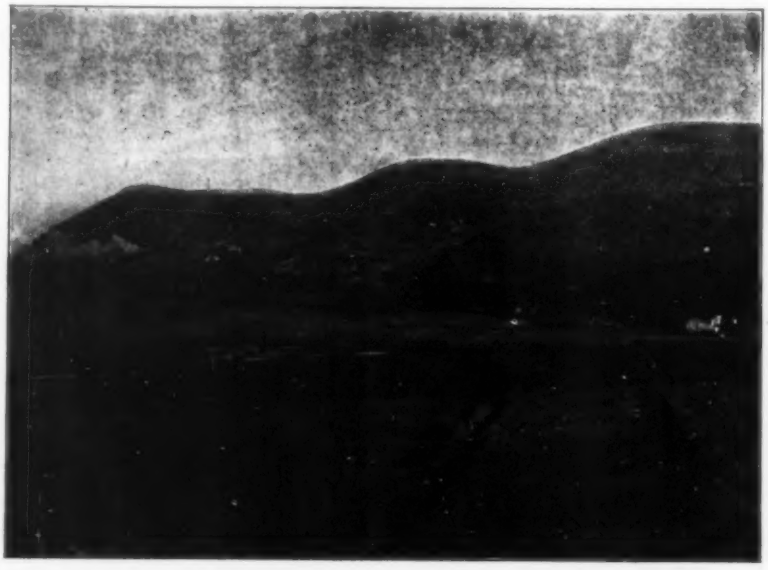


FIG. 1.—THE SURFACE OF THE HEAPS ARRANGED FOR WASHING.

Successful heap-washing depends on the efficient ventilation of the mass, the trouble usually being a too great excess of "fines" produced in mining the ore, which cement hard and clog up the air passages.

The copper liquor as it runs from the heap contains some ferric iron in solution, which is very objectionable. In order to remove the ferric iron the liquor is run over a smaller heap of fresh mineral known as a "filter bed" which reduces the ferric iron. This "bed" is laid inside a reservoir formed by a masonry dam across a small ravine, and the liquor, after percolating through the mineral, remains in contact with it until it is required to be drawn off to the precipitating tanks. When the mineral is fresh, the reduction of the ferric iron takes place rapidly, due to the Cu_2S as shown by equation (3), but the iron pyrite itself has an effective reducing action on ferric iron in solution according to the following equation:

(4) $7Fe_2(SO_4)_3 + FeS_2 + 8H_2O = 15FeSO_4 + 8H_2SO_4$.

A general view of the precipitating tanks and the masonry dam forming the reservoir for the percolation of the copper liquor is shown in Fig. 2. The principal constituents of the liquor as it enters the cementation tanks are as follows, the figures given representing the grams per cu. m. or units per million parts: Copper, 4,000; ferric iron, 1,000; ferrous iron, 20,000; free sulphuric acid, 10,000; and arsenic, 300. The large quantities of ferrous iron and free sulphuric acid are due to the fact that the waste liquor from the cementation tanks after the copper has been precipitated ("salida" liquor as it is called), is pumped back and used for washing the heaps in addition to fresh water, and consequently these solutions tend to become concentrated. The liquor is then run from the reservoirs at about 300 cu. m. per hour through the precipitation tanks over pig iron in order to precipitate the copper in the form of so-called "cement copper" or "copper precipitate." These cementation tanks are arranged in series on the slope of a hill, the liquor passing backwards and forwards until it is discharged from the lowest tank of the series free from copper.

FIG. 2.—PRECIPITATING TANKS AND THE MASONRY DAM FORMING RESERVOIR FOR THE PERCOLATION OF THE COPPER LIQUOR.

Each series consists of three tanks in parallel arranged so that the liquor can be divided up and passed along as many tanks as necessary, depending on the quantity of liquor that is being run through and on the varying temperature of the liquor with different seasons; the hotter the solution, which in summer reaches 100° F., the faster the rate of precipitation. Each tank is about 320 feet long, 5.5 feet wide, by 2.25 feet deep, and has a slope varying from 2 per 1,000 in the first series to 11 per 1,000 in the last, the reason for the increase in slope being, that as the liquor becomes impoverished in copper the free acid present is more active in wastefully dissolving the pig iron, an action which is considerably diminished by increasing the velocity of the liquor by means of the increased slope of the tanks. The tanks themselves are made of 9-inch by 3-inch boards attached to wooden frames set in cement constituting a wall supporting the sides of the tanks. A view of the tanks showing the method of removing the cement copper is shown in Fig. 3.

FIG. 3.—METHOD OF REMOVING THE CEMENT COPPER.

No metal is used in the construction, hard wooden pegs being employed to attach the boards to the frames. The spaces between the boards are carefully calked with oakum and pitch in order to render the tank water-tight. At each end of the tank is an arrangement by which a door can be dropped in and luted so as to cut out that particular tank, and there are also wooden plugs that can be removed so that the liquor from that tank can be run off, thus allowing for the

removal of the precipitated copper. A few old boards are placed on the bottom of the tanks for their protection, and on these are piled up the pigs of iron which are laid across the tank at the bottom, the next layer being at right angles to the first, and so on until the tank is filled; each foot-length of the tank contains about one ton of iron.

The liquor is allowed to run through the system of tanks and needs no attention except to remove the precipitated copper and to add fresh iron. The "salida" liquor, containing from 15 to 20 grams of copper per cu. m., is allowed to run to waste, for the reason that about this copper-content the amount of iron required to precipitate the copper equals the value of the copper recovered. Daily some of the tanks are cleaned out by being closed as described above, the liquor meanwhile passing down the other tanks of the series; the liquor is run off into settling tanks, any copper in suspension being there recovered; all of the iron is removed from the tank and piled on to the dividing wall, at the same time the copper adhering to the iron is knocked off and thrown back into the tank.

The dirty-looking precipitate is then transferred to the cleaning and concentrating plant, the iron is replaced in the tank and the liquor again allowed to run through it. This crude precipitate, containing about 70 per cent copper, is thrown, a little at a time, on to a perforated copper plate placed at the head of a long launder or tank and is washed through the plate by a strong stream of water from a small nozzle. The material that does not pass through the screen consists of leaf-copper and small pieces of iron; this material is thrown into a heap and afterwards sorted over by girls who remove the pieces of iron. The precipitate that passes into the launder is repeatedly turned over against the stream of water, and by this simple means a concentration is effected.

The first few yards of the launder contain a red precipitate known as "No. 1 precipitate," containing 94 per cent copper and less than 0.3 per cent arsenic; following this is "No. 2 precipitate," containing 92 per cent copper and between 0.3 and 0.75 per cent arsenic, while below is the "No. 3 precipitate;" this is in a state of very fine division and contains on an average 50 per cent copper and 5 per cent arsenic. This last-named portion, which carries all the graphite from the pig, contains the bulk of the antimony and bismuth that is also precipitated from the liquors.

Classes Nos. 1 and 2 are removed to the drying sheds and are bagged for shipment to the refinery; the No. 3 precipitate is removed, moistened with acid liquors, made into balls by hand, and dried in the sun. These balls become cemented hard and can be readily transferred to the smelter, where they form part of the charge for the blast-furnaces and are run down to matte to be subsequently bessemerized, thus effectively removing the arsenic, antimony and bismuth that they contain.

The reactions that take place in the cementation tanks are given in equations 5, 6 and 7. The first reaction that occurs in the liquor running over metallic iron is the reduction of the ferric sulphate to ferrous sulphate, the final result being in accordance with the equation:

(5) $Fe_2(SO_4)_3 + Fe = 3FeSO_4$.

This action causes the consumption of the pig iron without any corresponding yield in copper, and consequently should be avoided as far as possible, by having all the iron in the ferrous state.

The second reaction is the precipitation of the metallic copper, brought about by electrochemical action. The iron becomes coated with copper, and thus the iron and copper in the acid liquor constitute a galvanic couple with a considerable difference of potential (or, in other words, a short-circuited galvanic cell.) It is due to the electrolytic action that the copper and all other metals present that are electro-negative to iron will be precipitated. (In the language of the electrolytic dissociation theory the reaction is due to the high solution tension of iron, compared with that of copper.) The ultimate action of the precipitation may be chemically expresed by the following equation:

(6) $Fe + CuSO_4 = FeSO_4 + Cu$.

Besides the reactions above mentioned there is one which causes the liberation of hydrogen, as evidenced by the bubbles of gas that may be observed to arise in the tank liquor. This action, which causes a wasteful consumption of the iron, may be expressed as a final result by the following equation:

(7) $Fe + H_2SO_4 = FeSO_4 + H_2$.

These three equations constitute the main reactions that take place in the precipitating tanks.

While the liquor is fairly strong in copper, the copper is mostly precipitated in a coherent form, but in the later stages as the liquor becomes impoverished it is precipitated in a powdery state—a condition which is more effective in its galvanic action with the iron, and thus unfortunately causes a larger precipitation of arsenic and other impurities than in the earlier stages. In the later stages also the "solution reaction" of iron in sulphuric acid, as given in equation (7), goes on to a proportionately greater extent than does the precipitation of copper, and consequently the cost of pig iron in precipitating the copper varies inversely as the quantity of copper in the liquor.

By keeping careful watch on the reduction as far as possible of the ferric iron before the liquor enters the tanks, and by giving it sufficient velocity through the tanks, a strongly acid liquor such as given above during a year's working will not consume more than 1.4 units of pig iron (containing 92 per cent iron) to one unit of copper precipitated. A valuable check on the iron being consumed can easily be kept by the laboratory, by analyzing the liquors before entering and after leaving the tanks, and from these analyses the quantity of iron that is being consumed can be calculated.

PALLADIUM.

At the general meeting of the American Philosophical Society held recently in Philadelphia, a paper was presented by Mr. Joseph Wharton, whose remarks are suggestive, taken in connection with the article of Dr. Wohlwill printed in our present issue. The following abstract of Mr. Wharton's paper is taken from a recent issue of *Science*.

The rare metal palladium belongs to the platinum group, yet in some respects resembles silver also. It usually occurs as a companion to platinum and thus exists in many places, but in such extremely small quantity that until lately the reworking of platiniferous residues from the various mines supplied most of what appeared.

The prevailing scarcity of platinum now directs attention to palladium as a practicable substitute; for it has many of the good qualities of platinum, while its price is rather lower. Both platinum and palladium occur in all the numerous nickel mines found among the Laurentian and Huronian rocks in the province of Ontario, Canada; the quantity of each of these metals varying from a mere trace to one or more ounces per ton, the average for each metal being one-hundredth of an ounce per ton.

Those Canadian ores carry nickel, copper, silver, gold, platinum, palladium, iridium and rhodium. The percentages of the precious metals are extremely minute and the various processes by which all these metals are recovered are naturally complicated and delicate; yet, as approximately 300,000 tons of these ores are treated annually by the Orford Copper Company, which owns most of those mines, that company now produces about 3,000 ounces of palladium annually. The reports of the Canadian government upon the metallic and mineral resources of the Dominion have never mentioned palladium as one of these resources. It is not known in what condition the palladium exists in that region, but as platinum

has been found there in the form of platinum arsenide, it seems probable that palladium may occur in the same condition.

Among the valuable characteristics of this metal are its hardness, ductility and malleability; it is also so non-corrodible that a polished sheet of it may long remain exposed to chlorine and hydrogen sulphide gases without tarnishing or losing its polish.

RECENT METALLURGICAL PATENTS.

COPPER.

E. P. Clark (patent 959,670, May 10) suggests a treatment of the mixed ores of copper by which the carbonates, oxides and metallic copper are first leached out and passed as copper sulphate solution into depositing tanks where the copper is deposited by electrolytic or chemical means. The portion of copper present as sulphide in the ore remains in the ore, since it is insoluble. The ore is then washed from the leaching tank and passed over concentrators. The sulphides being the heavier part are easily separated and are then oxidized by roasting and added to the next charge of ore in one of the leaching tanks. It is thus only necessary to roast the actual sulphides contained in the ore and not the entire body of ore. The solution in the leaching tank is prepared as follows: Sulphuric acid is added to water until the gravity reaches 7½° to 8° Baume; sulphate of iron is added in the proportion of 1 to 1¼ per cent of the total weight of acid and water. This solution is passed into the leaching tank on top of the ore and filters through it slowly, coming away from the filter bottom as a blue solution of copper sulphate. In connection with this we may refer to the article on wet methods of extracting copper, published on another page of this issue.

L. E. Porter (patent 759,220, May 3) intends to increase the scope of the cyanide process. He first subjects the ore in the usual way to the action of an alkaline cyanide solution, thereby dissolving gold, silver, copper, etc. Gold and silver are precipitated by the ordinary method with zinc shavings. The resulting cyanide solution is treated with an acid, yielding copper cyanide precipitate and hydrocyanic acid which are separated from each other. The hydrocyanic acid solution is neutralized with an alkaline oxide, such as caustic soda, caustic lime or caustic potash, thereby producing an alkaline cyanide solution suitable for the repetition of the process described before. The cyanide copper precipitate is mixed with an alkaline solution, and the mixture is treated by electrolysis, whereby metallic copper is deposited and an alkaline cyanide solution obtained, suitable for the further treatment of ores.

GOLD.

Besides the last patent noticed above under copper, there was granted a patent to I. Anderson (759,493, May 10) for the treatment of waters flowing from mines or stamp mills, etc., and containing gold and silver ores in which copper or iron are also present. For the deposition of the precious metals, the treatment of the solution with lime in finely divided condition is recommended. In case the water does not contain copper or iron in addition to gold and silver, it is recommended to add either copper or iron or both to the solution before adding lime. Gold and silver are stated to be deposited in solid form and may be recovered by filtration.

ALUMINIUM.

To the long list of aluminium alloys, which are already more or less successfully used, another is added in a United States patent granted to S. Kneppel (patent 759,617, May 10). It consists of 6 parts of aluminium, 1 part of zinc and 2 parts of Babbit metal, by weight, the Babbit metal consisting of 96 parts tin, 8 parts antimony and 4 parts copper, by weight. This alloy is claimed to be suitable for the manufacture of vats, coolers, kegs, etc., for the manufacture, storage and transportation of lager beer.

CHROMIUM.

A. K. Eaton (patent 759,424, May 10) proposes the manufacture of pure chromium by the following method: Sodium bichromate and zinc sulphate are thoroughly mixed and subjected to a red heat, resulting, besides the development of oxygen, in the formation of zinc chromite and sodium sulphate.

The sodium sulphate is removed by washing with water from which the dissolved salt may be recovered by evaporation and crystallization. The chromite of zinc thus obtained in form of a gray powder, is reduced. It is mixed with powdered carbon, moistened slightly with water containing a little molasses, sugar or other binder, and formed into very solid, compact cakes by heavy hydraulic pressure. These cakes are placed in a suitable crucible or furnace, and are heated until the zinc is thoroughly volatilized, while the carbon combines with the oxygen of the chromite, leaving pure chromium in lumps.

IRON.

W. M. Brown and D. Reynolds (759,590, May 10) suggest the following process of producing iron or steel direct from ores. Granulated iron oxide ore is mixed with sufficient granulated carbon to deoxidize the ore and carbonize the iron to the extent required for the production of the iron or steel desired. This mixture is introduced into revoluble crucibles set in a revoluble shell or furnace. This shell or furnace is heated by the waste products of combustion coming from the melting or fusing furnace. During the revolution of the crucibles the mixture of iron oxide and carbon is thoroughly mixed, but during this process the mixture in the crucibles is protected from the direct action of the products of combustion used in heating the crucibles. After deoxidation and carbonization have taken place, sufficient flux is introduced into the mixture to remove impurities, and the mixture is heated until melting or fusion of the iron results.

T. Rouse (758,853, May 3) proposes to convert iron-sand concentrates into briquettes or lumps as follows. The sand is thoroughly mixed with a solution of about one measure of commercial water-glass in 50 measures of hot water. The quantity of the water-glass solution must be such that when cooled the mixed mass is in a stiff plastic condition, although gritty, so that it can be moulded into briquettes, which are pressed and conveyed to a drying chamber, in which they are hardened in the presence of hot air and steam.

W. W. Hout (759,526, May 10) proposes a welding flux, consisting of calcined borax, iron or steel filings, drillings or cuttings, furnace dross, slag or cinder, and calcined marble dust, in certain proportions.

GENERAL METALLURGICAL APPARATUS.

In order to obtain metals from their ores by chemical or electrochemical means, with a wet process, the ore must undergo previously purely metallurgical treatment, in which the leaching operation is of great importance. Any improvement in mechanical details of this operation helps, of course, the whole process. G. W. Stead (758,713, May 3) describes an apparatus in which at the same time that a liquid is agitated the ore fed in at one end of the receptacle is carried forward progressively and brought into intimate contact with the liquid. For this purpose the pan is provided with a series of agitators or scrapers which extend transversely across the pan with their lower ends adjacent to the bottom. These scrapers are hinged so that they can swing in one direction, and a reciprocatory movement is imparted to the scrapers, so that the ore fed in at the rear end of the pan is advanced progressively through the liquid by the several scrapers in their forward reciprocation and automatically discharged from the latter by the action of the forward scraper.

E. C. Wills (759,791, May 10) patents a furnace for the reduction of ore and for the melting of pig iron and their

metals. His purpose is to provide hearth-like surfaces down which the ore and metal may feed by gravity and to subject the material so fed to the action of a flame from an oil burner independently of or in conjunction with the heat from an ordinary furnace.

R. Baggeley and O. S. Garretson (760,172, May 17) describe a furnace for utilizing the heat contained in molten slag to heat a steam boiler, the molten slag being carried past the heating surface of the boiler by an endless flexible carrier.

F. Siemens (760,263, May 17) describes a regenerative gas furnace with gas producers in close proximity to the furnace chamber; usually the gas is evolved from the producers at a comparatively low temperature, and it is necessary to preheat the gas. Means for this purpose are described.

A. P. Gaines (759,171, May 3) describes a hot-blast apparatus for metallurgical furnaces in which the combustion chamber is entirely separate and independent from the stove or regenerative chamber, so that when the blast is put on the cold-air blast can pass only through the stove and never through the combustion chamber.

G. K. Hamfeldt (759,991, May 17) describes a construction of blast-furnace tops, designed to provide for a good distribution of the stock, especially where a considerable amount of mesaba or other fine ores is used. For this purpose a metal shell is provided which is supported on the stack, and which is of smaller diameter than the internal diameter of the stack; this shell acts as a deflector to propely distribute the stock.

I. R. Davis (758,622, May 3) describes a reversible door for furnaces with details of construction for opening and closing the same.

A. C. Kloman (759,449, May 10) describes a device, which, if a leak occurs in twyers, bosh-plates and cooling boxes, enables one to detect which one of the cooling devices is leaking.

E. E. Taber (759,768, May 10) describes an improved twyer attachment for blacksmith's fords.

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS.

By George P. Scholl, Ph. D.

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS.

Electric Furnaces and Furnace Products.

Process of Producing Carbide.—W. S. Horry, Niagara Falls, assignor to Union Carbide Co. Patent 757,617, Apirl 19, 1904. Application filed March 29, 1902.

The process applies specially to the production of calcium carbide. The carbide-forming materials, which act as resistance-conductor, are subjected to the action of the current between electrodes so disposed that the current density through the charge increases gradually, till the carbide is formed and brought to a molten condition. A suitable form of furnace for carrying out the process consists of a cylindrical vessel of refractory material, the lower part of which is provided with a number of cylindrical electrodes. The latter are disposed radially, two rows of eight each being shown in the drawing. All these electrodes are connected to one pole of a source of electric current, either direct or alternating. The other pole is connected to a single electrode, situated in the center of the bottom of the furnace, a tap hole for the molten material being arranged close to it. The lines of current flowing thus converge from the various upper electrodes to the single central electrode, thus giving a gradually increasing current density. The water gases pass up through the charge and serve to preheat it, finally escaping near the top of the furnace. Instead of employing a charge containing large pieces of coke, which by their contact with each other afford paths for the current, a mixture of ground coke and lime may be used. In the latter case, however, either an initial conducting path has to be provided, or the radial electrodes must be arranged nearly in the same horizontal plane as the top of the central electrode, the charge being preheated by conduction and radiation, so that it acts as resistance conductor when it reaches the paths of current flow.

Process of Electric Heating.—W. S. Horry, Niagara Falls, asignor to Union Carbide Co. Patent 757,618, April 19, 1904. Application filed April 5, 1902.

The process relates to the heating of a body of conducting material, especially a mixture of a metallic compound and a reducing agent, and more specifically of lime and carbon for the production of calcium carbide. It is based on the use of polyphase currents, preferably with many phases, the idea being to so distribute the paths of the various phases within the charge, which acts a resistance conductor, as to effect an even heating. The furnace used for this purpose, designed for a current of 2000 kw. or more, is a cylindrical vessel near the lower portion of which are arranged two horizontal rows of cylindrical carbon rods, radially disposed, which serve as electrodes. Each row is made up of 24 electrodes, each of which is connected to one terminal of a dynamo or transformer furnishing a current of twelve phases. A tap hole for the molten product is provided at the bottom of the furnace, the process being intended for continuous working. The use of a large number of electrodes is a desirable feature in the process, inasmuch as relatively small electrodes can be used without unduly heating them or their connections. Practically the maximum voltage of the generator is said to be always maintained between one pair of electrodes, the working current thus being always at the maximum furnished by the generator instead of varying from zero to the maximum as with single-phase current. For this reason a saving of about 30 per cent by weight of electrodes is claimed to be effected, as the total cross-section of the electrodes may be that much smaller than would be necessary for a furnace of the same capacity using single-phase current.

Process of Electric Heating.—W. S. Horry, Niagara Falls, assignor to Union Carbide Co. Patent 757,619, April 19, 1904. Application filed April 28, 1902.

This process, like the preceding, depends upon the use of polyphase currents for heating a body of conducting material, especially a mixture of lime and carbon for the production of calcium carbide. It differs from the preceding in so far as one terminal of the various phases of the dynamo is connected to individual carbon electrodes, while the other terminal of each phase leads to a common, centrally situated electrode. The furnace is shown in Fig. 1. The top is closed by a charging device, consisting of bell and hopper, while an outlet tube carries off the waste gases. Two series of electrodes, each containing eight of them, numbered 1 to 8, are introduced through the sides. The armature of the dynamo has eight separate windings, one terminal of each of them being connected to the respective electrodes. The other phase terminals lead to the common electrode 9, which is provided with a tap hole. The process is intended to be continuous.

Method of Electric Heating.—W. S. Horry, Niagara Falls, assignor to Union Carbide Co. Patent 757,620, April 19, 1904. Application filed June 6, 1902.

The method relates principally to the production of calcium carbide and depends upon maintaining a conductive body in a molten state between the electrodes, the body being so disposed that the current density of the current passing through it is variable, and generally a maximum at or near the tapping hole. Thus the body is heated at that place to a relatively high temperature and may be readily withdrawn in a molten condition from the furnace. Such conductive body may be calcium carbide, iron, carbide of iron, or other suitable material. Various forms of apparatus, in which the method may be carried out, are shown in the specification. One construction shows a furnace, the side walls of which converge to a contracted portion in the middle, at which point is located one

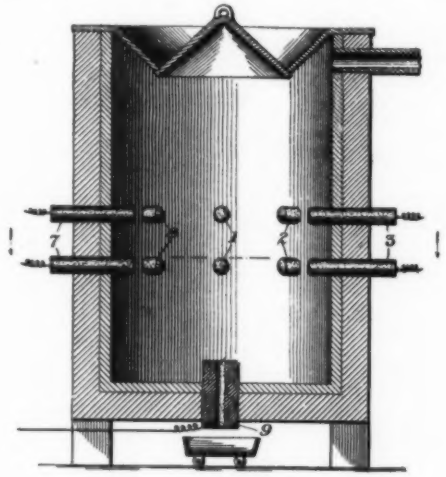


FIG. 1.—CALCIUM CARBIDE FURNACE.

electrode and the tapping hole. Two electrodes of opposite polarity to the central one are located, one at each of the wide ends of the furnace, the bottom of the latter being horizontal. As the cross-section is thus narrowest near the tapping hole, the temperature of the molten material will be the highest, so that it can easily be tapped out. Another construction is shown in which the bottom of the furnace slopes up to a point in the middle, where the tapping hole is situated. The side walls in this case may be straight or also may converge toward the middle. The central electrode may also be omitted, the electrodes at the ends being made of opposite polarity. Several more modifications of the construction are shown in the specification.

Electric Furnace.—W. S. Horry and Edgar F. Price, Niagara Falls, assignors to Union Carbide Co. Patent 757,621, April 19, 1904. Application filed October 11, 1902.

This specification, issued to the above inventors, is substantially worded like the preceding one, issued to one of them, and covers the designs of the furnaces described there, the constructions shown in the drawings being exactly the same.

Electric Heating.—E. F. Price, Niagara Falls, assignor to Union Carbide Co. Patent 757,633, April 19, 1904. Application filed April 5, 1902.

This process also applies primarily to the production of calcium carbide, and the leading idea is the increasing of the density of the electric current through the material under treatment towards the point where it is discharged, the material serving again as resistance conductor. This purpose is sought to be accomplished by supplying additional current to the material at various points on its way to the tapping hole. A furnace construction embodying this idea is shown in Fig. 2. The furnace proper consists of a vertical stack and back of refractory, non-conducting material, and a hearth of refractory conducting material, such as carbon. The latter is connected to one terminal of a source of current. A tubular body 8 of resistance material, preferably in a loose condition, such as lumps of graphitized carbon or coke, is placed into the furnace. The material to be treated is disposed in the form of a column 9. A series of electrodes of opposite polarity to the carbon hearth dip into the top of the resistance conductor. The current density in the lower part of the resistance conductor, and thereby the temperature, is increased

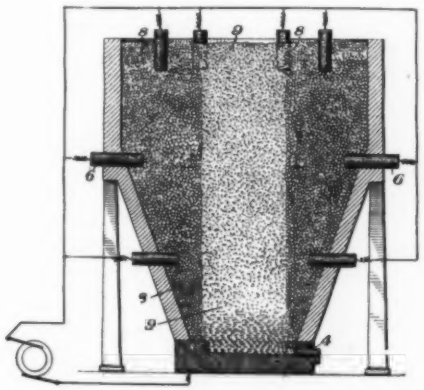


FIG. 2.—CALCIUM CARBIDE FURNACE.

in two ways, in the first place on account of the decrease in cross-section of the resistance conductor, and in the second place by supplying additional current through radial electrodes. As the lower part of the charge column 9 is brought to a sufficient temperature, the calcium carbide is formed and may be drawn off through the tapping hole 4. The furnace is intended to run continuously. Another construction shows the principle outlined above applied to a horizontal furnace.

Electric Resistance Furnace.—E. F. Price, Niagara Falls, assignor to Union Carbide Co. Patent 757,634, April 19, 1904. Application filed April 30, 1903.

The furnace constructions shown in this specification also embody the idea of the superposition of currents in order to increase the density of the current in the resistance conductor, which serves to heat the material to be treated, so as to subject the latter to increasing temperatures on its way to the tapping hole, which is preferably placed adjacent to the region of maximum current density. One of the present constructions shows a cylindrical furnace of refractory material, provided with bell and hopper, and outlet for waste gases at the top. It has a lining of carbon, which serves as the resistance conductor. A hearth of carbon, in contact with the carbon lining, is connected to one pole of the electric current. Three hollow metallic rings surround the carbon lining, and are in close contact with it. They are cooled by water or another cooling medium. The first ring is arranged at the top of the furnace, the second one somewhat above the middle, and the third one a little distance above the hearth. All three of them are connected to the other terminal of the current. The density of the current flowing through the carbon lining is thus increased by steps from above downwards, and the temperature of the lining is raised accordingly. A second construction shows a horizontal furnace embodying substantially the same principles. In the third construction the furnace has the form of an inverted cone of refractory material, surrounded by a water-jacket. The hearth is again of carbon, as in the above construction, and connected to one pole of the current. The resistance conductor in this case is formed of a tubular body of carbon, which rests in the middle

of the hearth and extends above the top of the furnace. At its upper end connection is made with the other pole of the current. A second current is superposed upon the first one in the lower half of the tubular resistance conductor, by the introduction into the hollow interior of the latter of another terminal of like polarity to the first. This is kept from contact with the upper portions of the interior of the tube by insulating rings, the contact only taking place at a point in the lower half of the tube. The tapping hole is arranged in the bottom of the furnace. The invention applies to the heating of materials, especially mixtures of a metallic compound and a reducing agent and specifically lime and carbon for the production of calcium carbide.

Process of Electrically Smelting Materials.—Alfred H. Cowles, Cleveland, Ohio. Patent 760,057, May 17, 1904. Application filed October 20, 1903.

The process consists in heating materials, the conductivity of which increases as their temperature rises, in an electric furnace provided with a casing of conductive material. The casing is cooled in order to form a non-conducting layer of solidified material upon it, and thus to prevent shunting of the current through the casing. The furnace consists of an upper cylindrical part, water cooled and carrying a bell and hopper, a middle part of frustio-conical shape, much longer than the upper part and also water cooled, and a lower part, formed by a hearth of carbon, connected to one terminal of the source of current. The upper part is divided from the middle part by a ring of refractory non-conducting material, through which a number of horizontal electrodes, arranged radially are introduced into the furnace. These electrodes are connected to the other terminal of the current. The hearth is also separated from the middle part of the furnace by a layer of refractory insulating material. On starting, after charging the furnace, *e. g.*, with a mixture of ground lime and coke for the production of carbide, the current is carried by one or more narrow cores of broken coke or carbon rods, extending from the upper electrodes to the hearth, until the surrounding portions of the charge are hot enough to serve as resistance conductor.

Process of Making Calcium Carbide.—Alfred H. Cowles, Cleveland, Ohio. Patent 760,312, May 17, 1904. Application filed June 28, 1902.

The process is intended for the continuous production of calcium carbide which is maintained in a molten condition in order to remove it from the furnace through a tapping hole. The charge is used as resistance conductor. The furnace used is illustrated in vertical central section in Fig. 3. It has a solid carbon hearth, surrounded by a layer of fire brick and a sheet-iron casing. A tapping hole is provided and a terminal, which connects the hearth with the source of electric current. An insulating ring of asbestos separates the hearth-casing from a dome-shape, water-jacketed sheet-iron structure, with water inlet and outlet. A pipe for carrying off waste gases and charge openings are also provided. The other electrode is made of carbon and suspended vertically in the furnace. It is intended to produce superheated molten carbide, which means that the carbide has not only been produced and melted by the heat from the resistance conductor but has also been raised to such a temperature that it will run through a tapping hole without requiring an additional supply of heat.

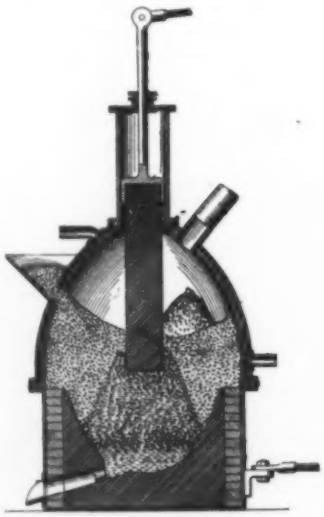


FIG. 3.—CALCIUM CARBIDE FURNACE.

Electrical Furnace.—R. M. Pelton, Detroit, Mich. Patent 759,909, May 17, 1904. Application filed December 3, 1902.

The furnace is intended for the use of dentists, and its essential part consists of a muffle in the walls of which a coil of heating wire of platinum is located. The outside of the muffle is surrounded by a further heating coil.

Electrolytic Production of Metals and Compounds.

Process of Electrolytically Extracting Copper and Zinc from Ores.—S. Laszezynski, Kielce, Russia. Patent 757,817, April 19, 1904. Application filed October 10, 1902.

The inventor states that in the electrolysis of a solution of sulphate of iron $FeSO_4$, at the cathode the bivalent ferro-ion is metallically deposited, while at the tri-valent ferri-ion ($Fe_2(SO_4)_3$) and that the iron, before being deposited from the latter compound on the cathode, has to be reduced again; in this manner there is soon set up a state of equilibrium, in which the same quantity of ferro-ions are reduced at the cathode as are produced at the anode. In the present process the detrimental side action at the anode is done away with by wrapping around the insoluble anode a cover or envelope of porous fabric. As soon as the current is turned on, a layer of pure sulphuric acid will form around the anode * * *, so that in a short time the anode is surrounded by diluted sulphuric acid free from iron * * *. In a state of equilibrium, the thickness of the envelop is in inverse proportion to the density of the current, as well as the free ions, also, the not dissociated molecules are diffused, but the latter are inactive and not to be taken into consideration. With a current density equal to one hundred amperes per square millimeter, and a concentration of the ferro-ions equalling half the normal, the thickness of the envelop is about three millimeters. In the strength of these observations, the inventor puts in the following claim: "Process for electrolytically obtaining metals, especially copper and zinc, out of their ores by means of insoluble anodes, consisting in tightly wrapping the insoluble anode in a porous and perfectly permeable envelop of fabric or other suitable material, the thickness of which is in inverse proportion to the applied density of current, for the purpose of preventing anodic oxidation of the cations."

Electrolytic Deposition Apparatus.—W. J. Jory and J. H. Jory, San Francisco. Patent 757,557, April 19, 1904. Application filed April 7, 1903.

The apparatus is intended for the deposition of a variety of metals, among them gold, platinum, silver and quick silver. It consists essentially of a trough-shaped box, of such a capacity that the solution under treatment shall flow through it in a hollow stream, while yet moving at such a rate as to carry off any sands contained in it. The bottom of the trough is formed of plain or corrugated plates of copper, silvered copper or other suitable metal, which constitute one electrode, generally the cathode. A wooden frame secured to the sides of the trough forms a support for a series of shafts, which work in bearings so arranged as to constitute receptacles for mercury, the employment of the latter being made for the purpose of lessening friction and forming an electric conductor. Electrodes of any suitable metal or carbon are fastened to these shafts and dip into the electrolyte. Means are provided by which the level of the apparatus may be changed to regulate the rate of flow as desired. One of the instances of application of the apparatus proposed by the inventor refers to the deposition of precious metals from the flow proceeding from stamp batteries. In this case they propose to introduce a soluble mercuric salt, such as mercuric nitrate or mercuric

chloride, into the batteries, "in such proportions with the metallic contents of the ore as will form an amalgam of such metals in so plastic a form that it may easily be removed from the plates, and yet of such firmness that it will not be carried off in the flow of water and sands." Upon electrolysis during the passage of the solution through the apparatus described above, the mercury is deposited on the cathodes of the apparatus, and according to the claim of the inventors all the precious metals contained in the flow are recovered by amalgamation with the mercury so deposited. Precise data as to voltage, current density, strength of solutions employed, etc., are conspicuous by their absence.

Apparatus for Circulating Liquids in Tanks.—G. E. Dunton, New York. Patent 758,430, April 26, 1904. Application filed September 15, 1903.

The apparatus is specially intended for the continuous circulation and thorough agitation of electrolytic solutions, in which deposition of metals takes place, in order to make the metal deposited at the cathode finer in grain and tougher; to shorten the time of deposition and keep the solutions from becoming heated. The means adopted for carrying out this purpose consist of a pair of centrifugal pumps situated near the top of one of the small ends of the tank and a system of piping which directs the liquid between the anodes and cathodes. The suction pipes of the two pumps reach down nearly to the bottom of the tank, so that the liquid is drawn downward.

Apparatus for Circulating Liquids in Tanks.—G. E. Dunton, New York. Patent 758,513, April 26, 1904. Original application filed September 15, 1903. Divided and refiled November 25, 1903.

The apparatus is adapted to the deposition of the copper shells used in electrotyping. The anodes and cathodes are arranged crosswise in the tank, and the two centrifugal pumps situated as above are each provided with one pipe running along the long side of the tank and furnished with a number of nozzles alternately disposed, so as to discharge the liquid towards the center of the tank between the electrodes and produce a compound motion of the solution.

Process of Obtaining Tin by Electrolysis.—E. Quintaine, Argenteuil, France. Reissued patent 12,214, April 26, 1904. Original, 699,012, April 29, 1902. Application for reissue filed April 20, 1903.

The bath used by the inventor consists of a solution of acid nitrate of tin to which a certain quantity of a salt of ammonia, preferably sulphate of ammonia, is added in order to neutralize it. For producing acid nitrate of tin, he prepares a solution of acid nitrate of tin, to which is added gradually a certain amount of salt of ammonia and a tin chloride, this gradual addition to the solution being continued until precipitation ceases and the bath becomes clear as water. The scrap is placed in a basket of wood or copper wire, and suspended in the bath as anode, while a cathode of lead is recommended. The tension of the current is given as ordinarily 1.7 volts, which should not be higher, and the current density as between 20 and 25 amperes per square meter.

Titanous Compound and Process of Making Same.—H. Spence, Manchester, England. Patent 758,710, May 3, 1904. Application filed September 11, 1902.

The invention relates to the manufacture of a new titanous sodium sulphate compound for use as reducing agent for various purposes, especially in the textile industry. The inventor has found that when a solution of the double basic compound of titanium sulphate and sodium sulphate, the preparation of which was described in his patent 670,819, of March 26, 1901, with the addition of sulphuric acid, is subjected to electrolysis, the resulting reduced solution contains a new double compound of titanous sulphate (sesquisulphate) and sodium sulphate of the composition (after drying in vacuo) Na_2SO_4 $Ti(SO_4)_2$ $5H_2O$. The solution is prepared by dissolving 100 parts by weight of the double basic compound of titanium sulphate and sodium sulphate in water, with the addition of 23 parts by weight of sulphuric acid containing 66 per cent of SO_3. The amount of water employed is such that the strength of the solution shall, after reduction, be of about 1.3 spec. gr. It may also be made by dissolving 100 parts of titanium hydrate containing about 20 per cent of TiO_2 in about 53 parts of sulphuric acid containing 66 per cent SO_3, diluting it to 1.3 spec. gr., and after subjecting it to electrolysis adding to the resulting reduced solution a little more than one molecule of sodium sulphate for every molecule of Ti_2O_3 present. The electrolytic reduction may be carried out in a lead-lined vessel, divided into two compartments by a diaphragm, the anode being of lead and the cathode of carbon, or also of lead, a solution of sulphuric acid in water being placed in the anode compartment. A current density of about 10 to 15 amperes per square foot of cathode surface, with about 4 volts pressure, is stated to give good results, electrolysis being carried out at a temperature between 15° and 30° C. The resulting titanous liquor is evaporated to about 1.45 spec. gr., and the mother liquor separated from the lilac-colored crystals which should be kept dry, in which condition they are said to be very stable. The new compound is said to be a powerful reducing agent and also very suitable for the production of other titanium compounds, *f. i.*, titanous acetate by double decomposition with sodium acetate. (See the note on page 250 on titanium trichloride.)

Electrolytic Apparatus.—H. S. Blackmore, Mount Vernon, N. Y. Patent 759,798, May 10, 1904. Application filed July 22, 1903.

The apparatus is especially intended for the electrolysis of an equeous solution of chloride of sodium, with a mercury cathode. It comprises a vessel for electrolysis and another one where the amalgam is discharged, together with several other vessels intended to regulate the flow of mercury. The electrolytic vessel contains a mercury cathode and a row of anodes suspended from the cover, a receptacle for solid sodium chloride being arranged on two sides. These receptacles are in communication with the electrolyte which they maintain in a saturated condition. The electrolytic cell communicates with two cylindrical vessels, which, in their turn, are connected to the amalgam discharging vessel. The latter is of rectangular shape, and the amalgam is introduced at one corner, whence it flows through several compartments formed by baffle plates alternately leaving passages at opposite ends, and issues at the corner diagonally across from where it entered. Electrodes of carbon or other relatively electronegative material may be employed for increasing the rate of oxidation of the sodium. The water in the discharging vessel flows in the opposite direction from the mercury. The mercury, after discharge, flows into a cylindrical vessel provided with a valve outlet controlled by a float, from which a pipe leads it into a collecting vessel, whence it is pumped back into the main reservoir. From the latter it goes to a cylindrical vessel with a ball float, which serves to maintain a column of mercury at uniform level in the electrolytic cell, and then enters the latter to be charged and discharged again as above. The movement of the mercury and amalgam through the various vessels, is said to be effected solely by the differences of level of the various bodies of mercury and amalgam, and these differences to be maintained by suitable predetermined adjustment of float-feed and float-controlled outlet and of the height of the connecting pipes.

Electrolytic Apparatus.—H. S. Blackmore, Mount Vernon, N. Y. Patent 759,799, May 10, 1904. Original application filed June 22, 1903. Divided and refiled February 23, 1904.

The apparatus is primarily intended for fused bath electrolysis with a view to the production of a lead sodium alloy and the recovery of the sodium therefrom. It is illustrated in Fig. 4. The electrolytic vessel proper 1 is made of iron heated by a burner 5 and connected with the negative terminal of the current. A cover supports an anode chamber, 14.

A disc 21 of iron wire gauze carries a diaphragm 22 of greater spec. gr. than the molten electrolyte, and lesser spec. gr. than the molten lead cathode. Magnetite broken into angular fragments, which pass through a 50-mesh has been found especially suitable. The anode is made of graphite while a cathode of molten lead fills the lower part of the outer iron vessel. The lead sodium alloy produced by electrolysis is said to be continuously displaced from the surface of the cathode beneath the diaphragm and to rise to the surface of the cathode around the anode chamber. Provision is made to remove the sodium from the alloy in the following manner. An annular pipe 29 with two rows of outwardly and downwardly opening perfora-

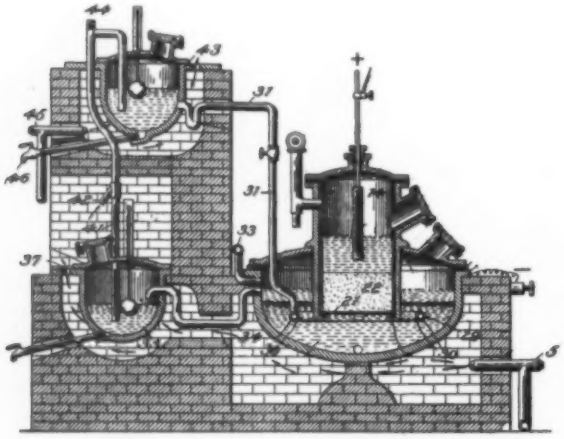


FIG. 4.—PRODUCTION OF SODIUM BY ELECTROLYSIS OF FUSED SALT.

tions is located in the molten lead cathode, through which a stream of molten sodium hydrate is introduced through pipe 31 from vessel 43. The molten sodium hydrate is said to be reduced by the sodium in the alloy to sodium oxide, with evolution of hydrogen, which latter escapes through pipe 33. The sodium oxide is said to float on the surface in a molten condition, and to be drawn off by an inverted syphon 34 into vessel 35. A layer of loose granular material, which may be small angular fragments of iron magnetite or ferrosilicon, is introduced beneath the surface of the cathode and kept submerged by a horizontal ring of wire gauze. This layer is stated to break up and distribute the streams of sodium hydrate rising from pipe 29 and bring them into intimate contact with the sodium in the alloy, the oxidation being facilitated by the innumerable local couples due to the contact of the sodium and the relatively negative particles of iron, etc. The sodium oxide in vessel 35 is kept molten by the waste gases escaping by passing along 34 to 35 and 37. The sodium oxide is preferably constantly converted into sodium hydrate, and for this purpose a pipe 41 leads from vessel 35 to vessel 43, past a steam injector 42, which serves both to introduce the water necessary for hydration and to carry the material upwards. The vessel 43 is also provided with a float level indicator and outlet 46, and it is heated by burner 45. The amount of sodium hydrate which is required to oxidize the sodium in the sodium lead alloy, continually passes from vessel 43 through a trapped outlet into the pipe 31.

Process of Manufacturing Peroxides.—F. Hinz, Berlin, Germany. Patent 759,887, May 17, 1904. Application filed November 23, 1903.

The process relates to the electrolytic production of magnesium peroxide and zinc peroxide in a vessel separated into two compartments by a porous diaphragm, with an anode of platinum or carbon, and a cathode of platinum or tin. The anodic electrolyte in the production of magnesium peroxide consists of a solution containing about 200 grams of magnesium chloride per liter while the cathode compartment is charged with a solution of hydrogen peroxide, containing about the same quantity of magnesium chloride, after any free acid has been neutralized with magnesium oxide or hydroxide. The voltage at the terminals is stated as about six to seven volts, the current density is not given. Magnesium peroxide in the form of a white powder is said to be deposited on the cathode, "when the current is sufficiently strong." It drops down into the cathode compartment and is collected, washed and dried at a moderate heat. The production of zinc peroxide takes place in a similar manner, but electrolysis is carried out at a pressure of 2.5 to 3 volts.

Apparatus for the Electrolytic Refining of Metals.—A. Schwarz, New York. Patent 760,023, May 17, 1904. Application filed November 22, 1902, renewed November 20, 1903.

The apparatus differs from the usual type in the arrangement of the electrodes. Instead of employing electrodes which are parallel to each other the inventor uses a large number of small cathode plates, suspended from a support which rests upon the top of the electrolytic tank in such a manner that their edges are turned towards the anode. The latter may be of the usual form, or it may be provided with projecting ribs, in order to give it also an increased surface. The inventor states that under conditions of practical copper refining he has found it practical to employ as many as forty-eight cathode plates, each twelve inches long and one inch wide, which at 24 square inches depositing surface per plate would give a total of 1152 square inches as compared with a single cathode plate of a total depositing surface of 288 square inches as heretofore employed with a corresponding section of the tank. To prevent the deposition of metal on the edge of the cathodes near the anode, the edge may be covered with some insulating material.

Storage Batteries.

Secondary Battery.—L. H. Flanders, Wilkinsburg, Pa., assignor to Westinghouse Machine Co. Patent 757,446, April 19, 1904. Application filed July 10, 1903.

The invention relates to means of supporting and separating the electrodes of storage batteries. The battery jar has a supporting block in the bottom at each end, upon which rest corrugated separator plates, preferably formed of porous earthenware. The edges of these separators are considerably thicker than the corrugated body portion, and fit tightly against each other, so as to completely enclose the electrodes, which rest on their upper edges by means of lugs. It is claimed that a battery of this kind may be conveniently and cheaply assembled, that the formation of short circuits is prevented, and that the electrodes may be readily removed without disturbing the separators.

Process of Making Storage Battery Plates.—Joseph Bijur, New York. Patent 757,718, April 19, 1904. Application filed February 10, 1903.

The plate consists of a number of grilles, which are assembled and united together by means of molten lead. In order that the grilles may not be clogged up by the molten metal, their interstices are filled with a material which can afterwards be dissolved by a suitable solvent. Such filling materials are soda ash or pearl ash or other bodies, which, under the conditions of the process, remain infusible, incombustible, non-delinquescent, beneficial or at least harmless, chemically, non-gas-giving and cohesive after drying from a paste made with a non-solvent, *f. i.*, alcohol, and soluble by an agent which does not injure the plate.

Storage Battery.—A. V. Meserole, New York. Patent 757,943, April 19, 1904. Application filed July 13, 1903.

The battery shows a containing vessel in which the positive electrode is supported from bars running across the top of the vessel, while the negative electrode rests on the bottom. The positive electrode is made up of a number of tablets or wafers of active material, which have a hole in their center and are

piled up, one on top of the other, on rods depending from the above bars. The negative electrode consists of a tray of bronze provided with partitions which form a series of troughs, in each of which a slab of active material is placed. The tablets of which the positive electrode is made up are composed of peroxide of lead, combined with about 1 per cent of chemically precipitated sulphur and about the same portion of a "chemical precipitate of silver compound—such, for example, as an oxide." These materials are mixed preferably with water containing an acid sulphate, the nature of which is not disclosed. The slabs of active material for the negative electrode are composed of lead, zinc, mercury and copper, combined in any convenient manner, but preferably in the form of chemically precipitated metal pressed into tablets. The quantity of zinc and mercury, by weight, should be about double that of the lead and copper. The electrolyte may be any liquid suitable for storage batteries, but with the above materials dilute sulphuric acid is preferably employed.

Storage Battery.—J. T. Niblett, Greenwich, England. Patent 759,007, May 3, 1904. Application filed February 5, 1903.

The cell consists of an outer vessel of lead, preferably forming the negative conductor, and a porous vessel placed inside of it. The porous cell contains a central conductor which is embedded in a packing of granular material, which fills the whole interior of the porous cell. The space outside of the porous vessel is also filled with granular material, constituting the negative element. The inventor finds it advantageous to use a mixture of two kinds of granules, one kind being formed of granulated lead or lead alloy, and the other of molded pellets of lead oxide, mixed with about 2 per cent of infusorial earth. The electrolyte fills the spaces between the granules.

Storage Battery.—W. J. Redmond, Cleveland, Ohio. Patent 758,692, May 3, 1904. Application filed September 29, 1902.

The plate is formed with a series of filaments or pin-like projections for the purpose of retaining the active material, about 250 filaments to the square inch having been found to give satisfactory results. The pins may be formed by inserting a suitable tool at a slight angle into the body of the plate, so as to shave or cut a thin filament from its face, at the same time leaving one end attached to the plate, and then turning the filament up or out, so that it stands at approximately right angles to the body of the plate. Strengthening ribs may be provided along the core of the plate by decreasing at suitable points the amount which the tool is allowed to penetrate into the plate.

Storage Battery.—V. G. Apple, Dayton, Ohio. Patent 759,058, May 3, 1904. Application filed June 5, 1901.

The battery is designed for use on automobiles. The inventor claims: "In a storage battery, a battery-containing conducting case, divided into two compartments, the inside surfaces of each compartment electrochemically active, a series of liquid-tight conducting cases, one within the other, and insulated one from the other, their respective interior and exterior surfaces electrochemically active, within each compartment, and a central element within the inner case of each compartment constituting the electrodes."

Method of Accumulating and Using Electrical Energy.—A. G. Betts, Troy, N. Y. Patent 759,065, May 3, 1904. Application filed August 17, 1903.

The inventor has found on passing an electric current of moderate strength through an electrolyte composed of a strong solution of lead fluosilicate containing some free acid, or of a somewhat acid solution of fluosilicates of lead and another metal as copper, with graphite electrodes, that peroxide of lead is deposited on the anode in a dense adherent and well-conducting form, while metallic lead or the other metal is deposited on the cathode, free fluosilicic acid remaining in solution. When the battery thus formed is allowed to discharge, the lead or other metal and the peroxide of lead recombine with the fluosilicic acid, thus restoring the original conditions. For charging the battery a current density of 10 to 20 amperes per square foot of negative electrode surface is suitable, the potential required being about 2 volts. A concentrated solution of lead fluosilicate at ordinary summer temperature contains about 900 grams of lead and 35 grams of about 2.38 spec. gr. fluosilicic acid per liter, about 20 pounds of this solution representing a storage capacity of 1 electrical hp. hour. It is stated that the lead during the discharge shows a tendency to become soft, so that at the end some of it may not be firmly attached to the electrode. If the charging current is reversed, the loosely attached pieces may be completely removed and a better adherence obtained for the deposit. The electrodes are therefore both made of insoluble material, so as to allow of a reversal of the charging current. A small amount of gas is given off from the positive electrode until the graphite has been evenly coated with peroxide of lead. On account of this a small amount of lead or other metal remains undissolved on the negative electrode after complete discharge, which amount, though small, would gradually accumulate. This is corrected by the introduction of air, which also helps in increasing the circulation of the electrolyte. It is stated that for certain purposes a solution of metallic salts of various acids, including fluorine acids, such as fluoboric, fluotitanic, fluosilicic acids and others, while carbon, platinum or other insoluble electrodes may be used.

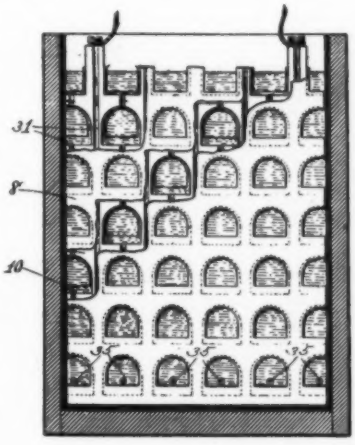


FIG. 5.—STORAGE BATTERY.

Electric Storage Battery.—A. G. Betts, Troy, N. Y. Patent 759,066, May 3, 1904. Application filed August 17, 1903.

The battery is illustrated in Fig. 5 and has been constructed with an idea to maintain any electro-deposited material which has become separated, in contact with the electrodes. The wooden cell is provided with a rubber lining. Each electrode is formed of a grating of artificial graphite with numerous openings to permit the free circulation of the electrolyte. Interposed between the electrodes are plates or mats of insulating material, such as rubber, which are provided with openings corresponding to those in the electrodes, but of smaller area. They thus form a pocket in the bottom of each electrode aperture, which catches electrodeposited material that has fallen off and hold it in contact with the electrode. Air nozzles with apertures 35 serve for the introduction of air.

Storage Battery Electrode.—C. J. Reed, Philadelphia. Patent 759,550, May 10, 1904. Application filed January 16, 1901.

The plate is of the Planté type and composed of a number of panels of thin metal, the exposed surfaces of which are angularly disposed with reference to the plane of the plate. The plate is preferably formed by passing molten lead between corrugated rolls of suitable surface configuration and dimensions, and then subjecting the plate thus produced to lateral edgewise compression for the purpose of reducing the corrugation-grooves to the smallest practicable width.

Electric Accumulator.—R. Fortun and E. Semprun, Madrid, Spain. Patent 759,431, May 10, 1904. Application filed September 30, 1901.

The invention relates to a process of manufacturing a so-called new active material for accumulators. The material is produced as follows: A sheet of lead and a lead tube are the respective electrodes in an electrolyte, and are subjected to a current, "the intensity of which at a pressure of 110 volts should not exceed 0.6 amperes for each 1,000 cubic centimeters of the liquid." The liquid is composed of "a solution of oxide of lead (PbO), chemically pure in a potassium hydroxide lixiviation at 32 per cent the oxide dissolved by saturation 700 parts, rock candy 200 parts, neutral potassium tartrate 100 parts." "As the current passes, a brown substance is slowly deposited on the lower sheet of lead, which substance when properly dried up and subjected to an electrolytic analysis during a sufficient time in order that in acting as a positive pole electrode within a platinum crucible its weight may not be further increased." One hundred grams of this substance are claimed to be capable of fixing 369 grams of oxygen. Then follows the announcement that this mysterious dynamatized substance "is not susceptible of being preserved and of reacting as already stated, except when immersed in an electrolytic liquid, such as a solution of seventeen parts of nitrate of lead in 100 of water acidulated with pure nitric acid until it registers 20° Bé."

Galvanic Elements.

Gas Battery.—J. H. Reid, Newark, N. J. Patent 757,637, April 19.

The battery described in the specification is illustrated in cross-section in the adjoining Fig. 6. A metallic pot preferably rectangular, has inside a number of longitudinal partitions 2, 2. A metallic extension 3 of the pot is insulated from the latter by insulating strips. The extension 3 has longitudinal slits forming a grid, the bars 7 of which are

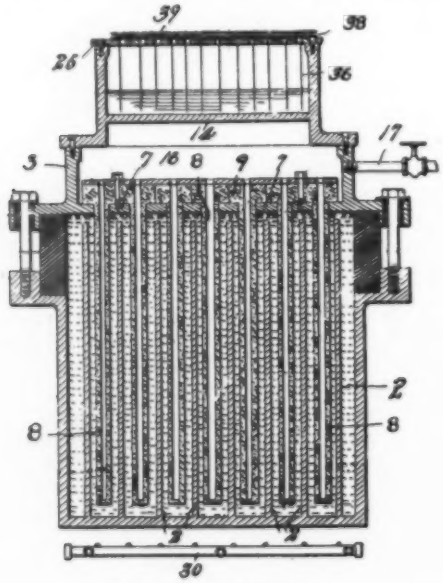


FIG. 6.—GAS CELL.

above the partitions 2; the electrolyte also extends above the latter partitions and is preferably composed of molten sodium or potassium hydrate at a temperature approximating 400° F. Between the bars 7 are inserted electrodes consisting of porous carbons, thin enough to pass between the bars, but having heads 9 which rest on the latter. The material of the electrodes is pervious to gases, but impervious to the electrolyte. Holes 8 are bored into the electrodes nearly to the bottom. An inclined flooring 14 of metal covers the gas chamber 16, which extends above all electrodes. A gas supply pipe 17 serves to introduce a combustible gas. The cover 25 of the cell is perforated, the follower plate having a number of fine holes bored close together. Wires 36 are suspended from the cover extending nearly down to cover 14. Above plate 25 is a layer of wire gauze 38 and finer gauze 39. One terminal of the current is connected to the electrodes 8, while the other is connected to the containing vessel. The electrolyte is heated by a suitable heater 30. The combustible gas passes into the porous electrodes and through them, and air is being forced into the electrolyte near one end. The electrolyte bubbles and froths over the flooring 14, the bubbles being broken by the wires 36 and the liquid returns to the main vessel, flowing down the inclined floor. Carbonic acid and other waste gases, released from the bubbles, pass off through the cover.

Battery.—J. Noble and E. L. Anderson, St. Louis, Missouri. Patent 759,740, May 10, 1904. Application filed June 1, 1903.

The battery consists of a lower part which contains two ring-shaped electrodes the anode being of aluminium and the cathode of carbon. The electrolyte is dilute nitric acid. The upper part of the cell is filled with a mass of carbon or coke, which is kept moistened, and has the purpose of condensing the gases given off in the operation of the cell. The operation of the cell is believed to be as follows: When the circuit is closed the current passes from the aluminium anode to the carbon cathode, decomposes water, setting free oxygen and hydrogen. The hydrogen is oxidized to water while nitrogen peroxide is formed. This gas, as the cell is maintained at a temperature of 150° F. or greater, oxidizes the carbon cathode, forming carbonic acid and nitrogen oxide. The latter unites with the oxygen given off at the anode, and passing into the condenser is there reduced to nitric acid and returns as such to the cell. The inventors, however, do not make any positive assertion on theories as to the chemical or electrochemical reaction which takes place in the operation of the cell.

Miscellaneous.

Apparatus for Electroplating Sound Records.—G. K. Cheney, New York. Patent 758,352, April 26, 1904. Application filed June 6, 1903.

The invention relates to means by which sound-record discs, made of soft wax are suspended in an electroplating tank and connected with the source of current.

Method of Generating Gaseous Mediums from Air.—J. N. Alsop, Owensboro, Ky. Patent 758,883, May 3, 1904. Application filed May 29, 1903.

The method consists essentially in the formation of an arc by allowing two electrodes, the lower one of which is stationary, while the upper one is movable, to come into contact and then drawing them apart. The electrodes are disposed in tubes, which are connected to an exhaust pump, the tubes being preferably arranged in sets of two or more, and operated alternately, the arc in one set being produced when the one in the other is extinguished. Mechanical means for bringing about this alternation are shown in the specification. It is claimed that a current of very low potential can be employed, as the electrodes are actually brought into contact. In order to meet the increased resistance as the electrodes are drawn further apart, a coil of high self-induction is introduced into the circuit. The inventor states that in practice he has used a dynamo designed to give 5 amperes at 500 volts, and an induction coil of a 100 ohms, and with the above apparatus he has produced a gaseous medium of a bleaching and purifying character with the voltage across the arc varying from 150 to 900 and the current in the arc circuit varying from 20 to 0.1 ampere. The gaseous medium produced is applied for whitening and purifying cereals.

Apparatus for Generating Gaseous Mediums from Air.—J. N. Alsop, Owensboro, Ky. Patent 758,884, May 3, 1904. Application filed May 29, 1903.

This specification covers the apparatus described in the preceding one by the same inventor.

Apparatus for the Treatment of Gases.—H. Pauling, Gelsenkirchen, Germany. Patent 758,775, May 3, 1904. Application filed November 20, 1903.

The invention relates to the treatment of gases, vapors and the like by electric spark discharges. The apparatus consists essentially of a funnel-shaped vessel, the funnel tube of which is connected with an air pump. A revolving shaft enters for some distance into the mouth of the funnel-shaped vessel, which is provided with an insulating cap at its ends. The cap is only pierced at one point by a laterally extending projection on the shaft. The latter is connected with one pole of an induction coil, while the funnel-shaped vessel is connected to the other pole. As the shaft revolves, a circular spark disc, caused by the continuous displacement of the above projection, will be formed, through which sparks the atmospheric air has to pass, as it is drawn through the apparatus by the air pump mentioned above. An alternativ form of the apparatus shows a hollow shaft and hollow projection, through which the entering air is drawn, which air consequently has to pass through the active zone, and is for some time subjected to the effect of the sparks.

Purifying Apparatus.—P. J. Boucher, Cleveland, Ohio. Patent 760,302, May 17, 1904. Application filed March 28, 1903.

The apparatus is intended for the purification of water or other liquids by electrolysis and consists essentially of a cylindrical receptacle with insulating head. A number of aluminium plates alternately connected to their respective electrode, are arranged longitudinally in the above vessel. A mechanical construction is shown which provides for automatically reversing the current at predetermined times, and for insuring the current being turned on or off, whenever the liquid is turned on or off. Means are also shown for automatically relieving the pressure of the gases which accumulate in the upper part of the electrolyzer during electrolysis.

SYNOPSIS OF ARTICLES IN OTHER JOURNALS.

A Summary of Articles on Electrochemistry and Allied Subjects Appearing in American and Foreign Periodicals.

INDUSTRIAL ELECTROCHEMISTRY.

Gravity Process for Electrolysis of Sodium Chloride.—The essential requirement of producing sodium hydroxide and chlorine by electrolysis of sodium chloride is to keep the anodic and cathodic products of electrolysis separated from each other. Various methods are in actual use to accomplish this result such as the use of a diaphragm cell or the use of a mercury cathode, both of which are worked on a large scale. The latest process which has made its appearance in the industry is the gravity process (Glocken process, as the Germans call it, on account of the form of the glocke, *i. e.*, bell, which surrounds the anode). This process has been worked since 1895 in Aussig in Austria, and has later been introduced in three plants in Germany. An interesting account of an investigation of this process is given by O. STEINER in the *Zeit. f. Elektrochemie* of May 6. The principle of the method is shown in Fig. 1. The anode is shown in the center and is surrounded by the glocke (bell). There are two openings in the top of the bell, one for introducing the chloride the other for carrying off the chlorine gas. The cathode is placed outside of the bell, and an overflow is provided by which the hydroxide formed at the cathode may flow off. The hydroxide mixed with chloride solution is in the outside cathodic compartment and in the lower part of the bell. Above this lower part, the bell contains a chloride solution with chlorine dissolved in it. The latter solution has a lower specific gravity than the hydroxide solution, so that it floats upon the latter. There is no diaphragm to keep both solutions separate from each other, and the separation is wholly accomplished by making use of the specific gravity. Between both solutions there is a layer of neutral solution and the main requirement for continuous operation is that this layer of neutral solution is maintained at its place during operation. It acts like a liquid diaphragm. When hydroxide is formed at the cathode, OH ions will, of course, begin to precipitate in the transmission of the current and will travel from the cathode to the anode. It is most essential that these OH ions should not come near the anode, since this would result in the formation of hypochlorite. The main cause which prevents them from passing through the neutral layer of solution which separates the anolyte from the catholyte, is the difference between the chloride concentrations at the anode and at the cathode. The author discusses this point at some length. He shows that the separating neutral layer always remains at the same place if the degree of alkali in the catholyte going off through the overflow is maintained constant by keeping the current constant and the rate of supply of chloride to the anode compartment also constant. If the current is increased or the rate of supply to the anode compartment is diminished, the contents of hydroxide in the catholyte increases and the neutral separating layer will raise upwards. In actual practice there should be a considerable distance between the anode and the lower rim of the bell, so that the neutral layer is at some distance from the anode, so that no ClO ions can reach the latter. The supply of fresh chloride should be evenly distributed over the whole anode surface. The author gives the results of a long series of experiments in which he first determined the efficiency which can be obtained. He found that in continuous operation an ampere-hour efficiency of 85 to 94 per cent may be obtained, yielding an hydroxide containing 120 to 130 grams KOH per liter, and 97 to 100 per cent pure chlorine gas, with a current density of 2 to 4 amperes per square decimeter of horizontal cross-section of the bell, with a voltage of 3.7 to 4.2 between the electrodes. The ampere-hour efficiency is nearly independent of the contents of hydroxide in the final solution, at least up to the maximum of 130 grams KOH per liter. The supply of fresh chloride should be uniformly distributed over the whole horizontal cross-section of the bell. If a colorless solution is wanted with the use of artificial carbon electrodes (such as made by Lessing), simultaneous sand filtration is necessary, since the carbon disintegrates. Without sand filtration the brown color may be removed by oxidizing agents; for in-

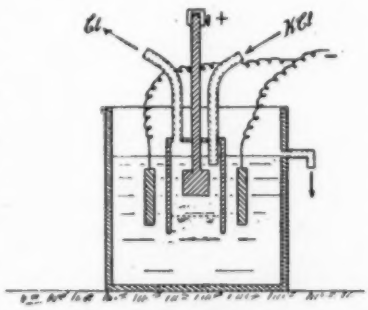


FIG. 1.—GRAVITY CELL.

stance, by means of chlorine gas. The distance of the neutral separating ozone from the lower rim of the bell is the greater, the smaller the contents of chloride in the supply of fresh solution and the smaller the current density per horizontal bell cross-section and the higher concentration of hydroxide in the final cathodic solution. The vertical distance of the lower surface of the anode from the lower rim of the bell is of great importance for the practical operation of the method. It should be such that the neutral solution is at least one centimeter below the anode. The above distance must therefore be the greater the higher the hydroxide concentration of

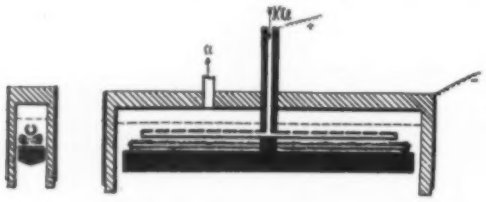


FIG. 2.—BELL FOR COMMERCIAL GRAVITY CELL.

the final solution and the lower the chloride concentration of the fresh supply of solution, and the lower the current density. For a 27.5 per cent KCl solution which is well distributed over the anode surface, in order to get a final solution containing 130 grams KOH per liter with a current density of 2 to 4 amperes per square decimeter of horizontal cross-section of the bell, the distance of the lower surface of the anode from the lower rim of the bell should be at least 4 to 5 centimeters. If these conditions are fulfilled, Acheson graphite is very suitable as anode material, since with a current density of 2 amperes per square decimeter, it does not disintegrate, the supply of solution containing at least 270 grams KCl per liter and being uniformly distributed over the whole anode surface and the hydroxide concentration of the final solution being not more than 125 grams KOH per liter. In an experiment with a current density even as high as 4 amperes per square decimeter, the author did not find a disintegration of Acheson graphite. For the technical operation the bells must be very narrow (about 10 cm.), since with greater breadth, the anode surface cannot be well utilized. Their length should be not more than consistent with a good distribution of the fresh solution over the anode surface. A bell, of the form shown in Fig. 2, 7 centimeters broad and 17 centimeters long could be used at 4.2 volts and 20 amperes, with a current density of 4 amperes per square decimeter. For a daily production of 5,000 kilograms solid caustic and 9,000 kilograms bleaching powder there would be required 5,000 such bells. With a current density of 2 amperes, 10,000 bells are required for the same output. It is estimated that the number of bells at present in use is 45,000.

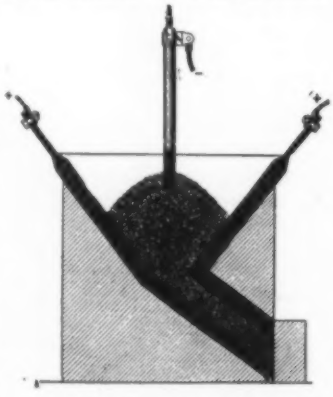


FIG. 3.—GRAPHITE FURNACE.

Manufacture of Artificial Graphite from Charcoal.—In *Metallurgie* of April 22, J. Weckbecker describes some laboratory experiments on the preparation of artificial graphite from charcoal. If carbides are heated beyond their dissociation temperature, the liberated carbon is obtained in the form of graphite. For technical applications it is important to choose a carbide which has a low dissociation temperature, while the other liberated element has a low temperature of vaporization, so that the graphite is obtained in pure form. As substances forming carbides he used aluminium oxide, clay and magnesia. In his first experiments he mixed powdered charcoal with varying quantities of aluminium oxide or magnesia and pressed this mixture to which tar had been added, to rods of 10 mm. diameter. To remove the tar, these rods were packed into a crucible, covered with charcoal and placed in a gas furnace. The rods thus prepared were used as resistances between two carbon poles, in order to bring them to different temperatures. With an addition of Al_2O_3, at a temperature of about 2,000° C. he obtained the best results if the amount of aluminium oxide was about 30 per cent. With lower amounts of aluminium oxide equally good results could be obtained if the temperature was increased. With the use of clay, instead of Al_2O_3, the formation of graphite was nearly as good, although there were small crystals of carborundum within the graphite. Experiments with magnesia, however, did not give satisfactory results. The reason is that magnesium carbide decomposes at red heat. The graphite formation with rods consisting of 20 per cent of magnesia and 80 per cent of charcoal was the same as with rods of charcoal without the addition of any magnesia. In other words magnesia gives no graphite, because at the reduction temperature of its oxide no carbide is obtained. The author thinks this is a proof that the carbide formation must precede the graphite formation. He describes various forms of furnaces which he has devised for continuous manufacture of graphite. He had some difficulty to find a satisfactory furnace lining; in some cases he used carborundum bricks. One of his furnaces is shown in Fig. 3; one electrode is formed by the carbon walls of the furnace, while the other electrode is perpendicularly suspended at the top.

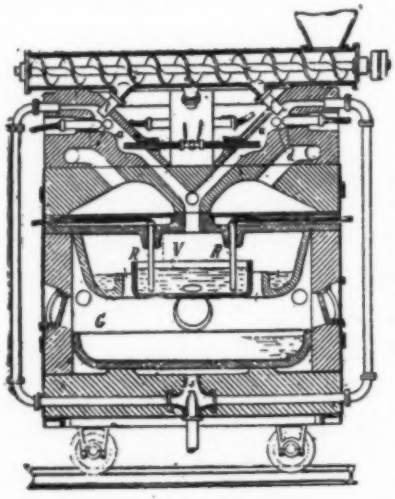


FIG. 4.—GLASS FURNACE.

Electric Glass Furnace.—The *Electrical Engineer*, of London, April 15, states that electrical furnaces are giving good results at the large glass factories at Matri in Tyrol, and at Plettenburg in Westphalia. Fig. 4 shows a furnace which is stated to have given satisfactory results. The raw material enters the top of the furnace through the tubes *tt*, and is then conducted in front of the arcs *aa*. The heat of the arc fuses the material which then drops into the receptacle V, where a current is passed through the material by means of the electrodes RR. The furnace is thus a combination of an arc and a resistance furnace. From V the molten glass descends into a crucible G, which is heated by an upward flame, formed by the gases which are developed during the fusion of the material. These gases are conducted to the flame through the lateral tubes shown in the illustration. In a later type of furnace the raw material, before having been exposed to the heat of the arc, is made into a paste with water. It is then pressed and dried, and finally it passes, in the shape of a ribbon, through the arcs between a number of electrodes, out of which it emerges completely fused. The heating is chiefly

done by radiation and care is taken to prevent impurities from the carbon entering the molten glass.

The Thermo-magnetic Selector.—An interesting application of magnetism to metallurgical purposes is the recent device of Sauveur and Whiting (*Proc. American Society for Testing Materials*, Vol. III., 1903.) If steel is rolled and finished too hot, it crystallizes while cooling and deteriorates in quality. If the finishing temperature is below 700° C., no crystallization ensues. But, this temperature is that at which iron becomes magnetic on cooling. In order, therefore, to select the steel rolled down to the proper temperature, a powerful electro-magnet acts upon the steel as it comes through the last pass of the rolls. If it is below 700° it is attracted downward, its end passes under a guide, and it passes off as an acceptable piece. If its temperature is above 700°, it is not attracted, and its end passes over the guide and the piece is thus rejected until rolled over at the proper temperature. It is an interesting application of scientific principles.

Cathode Copper.—An editorial is published in the May issue of *Metal Industry* claiming that cathode copper received from electrolytic refineries often greatly lacks in uniformity; the reason is thought to be that the electrolytic refining process is exceedingly delicate and unless constant vigilance is maintained concerning several points, the quality of the cathode will suffer. The writer advocates the use of ingot copper in preference to cathode copper.

Electric Furnace Processes.—In *Cassier's Magazine* of May, J. WRIGHT gives a description of various electric furnace processes which have either been proposed or have been introduced into practice. The article is fully illustrated.

THEORETICAL AND EXPERIMENTAL.

Electrolysis of Sulphites.—The *Zeit. f. Elektrochemie* of April 22, contains a long paper by A. FRIESSNER giving the results of a laboratory investigation. During the electrolysis of neutral and alkaline sulphites of potassium or sodium, there is no development of oxygen at the anode for an average current density, but the sulphite is oxidized to sulphate. Under suitable conditions dithionate may be formed according to the equation

$$2SO_3'' + 2OH' + 2H\cdot + 2F = S_2O_6'' + 2H_2O$$

where F means a positive charge. In an acidulated solution dithionate is never obtained, but sulphate is formed exclusively. If bisulphite is electrolyzed without a diaphragm, sulphate is formed at the anode and hydrosulphite at the cathode, and there is also formed thiosulphate, probably by cahodic reaction. The electroiytic formation of dithionate occurs at ordinary temperature at an anodic potential which is higher than that required for the formation of sulphate. This potential required for the formation of dithionate can be best obained by a previous anodic polarization of platinized anodes in sodium hydroxide. The increase of temperature favors the formation of dithionate. The electrochemical formation of dithionate may be compared with that of persulphates and percarbonates.

Electrolysis of Water with Alternating Currents.—R. G. VAN NAME and L. GRAEFENBERG have investigated the formation of oxygen and hydrogen gases by electrolysis of water with alternating currents; their results are published in an article in the *Zeit. f. Elektrochemie* of April 29. They measured the output of the gases and the voltage required for platinum and gold electrodes in sulphuric acid, and for silver, nickel or copper electrodes in potassium hydroxide. The frequency was kept constant at 55 periods per second, while the current density was varied. The highest efficiency which was observed was only about one-half that which can be obtained with direct current. Moreover, on account of disintegration of the electrodes, the maximum obtainable efficiency could be maintained only for a limited time. Various materials were examined concerning their suitability for electrodes with alternating currents, but it seems that a substance which is a good conductor and which does not integrate at a high frequency does not exist.

Calomel Electrode.—A paper published in the *Zeit. f. Elektrochemie* of April 29, gives the results of an investigation by H. LEY and C. HEIMBUCHER on the concentration of mercury ions in the calomel electrode and on the solubility of calomel. For the $Hg_2^{\cdot\cdot}$ ionic concentration in a 0.1 normal electrode and in the 1 normal electrode they find the values, 2.0×10^{-18} and 3.5×10^{-18}, respectively.

Concentration of H· ions.—The *Zeit. f. Elektrochemie* of May 13, contains a paper by E. SALM on the determination of the concentration of H· ions in a solution with the aid of indicators; with some polemical remarks concerning an investigation of Sallesky. For hydrogen electrodes he used platinum and palladium sheets and found that palladinizing the electrodes was more useful than platinizing, since with the former a better constancy is obtained, and the equilibrium is reached more quickly. To cover the electrode with palladium sponge he uses the following solution which is quite analogous to that of Lummer and Kurlbaum for platinizing; one gram of palladium chloride is dissolved in 30 grams of water, and 5 per cent of platinizing liquid is added as a catalyzer.

BATTERIES.

Lead Accumulators for High Discharge Rates.—In the *Electrical World and Engineer* of May 21, W. W. DONALDSON discusses storage battery plate construction with special reference to high discharge characteristics. It is not only in automobile work that high discharge rates are required. The largest field in stationary work where abnormal discharges are used, is where batteries take the peaks of rapidly fluctuating currents, as in tramway stations. The energy of inventors is therefore now directed toward producing plates in which the "time limit of capacity" is small and inversely, the "rate limit of capacity" is large. At present the so-called high discharge rate is four times the normal or eight-hour rate, and the ampere-hour efficiency is hereby diminished to 42 to 58 per cent of the normal capacity. The best plate is the one which, towards the end of its useful period goes to pieces very rapidly and loses its mechanical and electrical efficiency simultaneously. It is the electrical characteristics which should have the first consideration in the design. "Provide for good electrical conditions and the mechanical will take care of themselves." The author offers as a fundamental requirement for both plates that their active material should be formed by an electrochemical action. No paste should enter into the construction of either plate. The positive grid (spongy lead plate) should be integral; no mechanical or autogenous joints should be present. No other metal save pure lead should be used. Castings of the basic plate should be avoided since hard-rolled lead gives a density and homogenity far superior to cast metal. A center web is necessary for the most even distribution of current. Rigidity, in excess of that required for proper elecrical conditions, need only be provided for in direct ratio to the total weight of the plate. Any allowance beyond this reduces the percentage of effective active support. The main conducting and supporting frame, which should never be counted on for any active formation, should not exceed 15 per cent of the weight of the plate proper, and this, with 10 per cent in addition (the residue of the active support) should be all that remains of lead when the plate has lost its useful capacity. The support area for the active material contact should be not less than 175 square inches per pound of plate proper, and should not be greater than 250 square inches per pound of support contact; that is, the thickness of the support contact after formation should not be less than 0.02 inch, which allows 0.01 as a conductor for each layer of active material. The author emphasizes the great importance of the requirement that the metallic contact of the active material support with the main conducing frame should be of such character as not to be impaired by any action due

to subsequent formation of active material. As much of the surface of the active material support as possible should be bathed by free electrolyte, and provision be made for vertical capillary diffusion, rather than for horizontal or through-and-through circulation. This active material support surface presented for discharge action should not be less than 33 square inches per ampere for a rate corresponding to 8 times the normal. The plate as a whole should be as thin as is consistent, and should not exceed ½ inch from face to face.

The author then discusses the lead peroxide plate. The active material must be in the form of a very thin layer, and should not exceed 0.008 inch in thickness, with a conducting backing 25 per cent greater or 0.01 inch thick. Such a layer would correspond to 0.8 of one cubic inch of peroxide per 100 square inches of contact support, or (allowing 2.1 ounces per cubic inch) 1.68 ounces per 100 square inches. This volume of peroxide at a capacity of 2.5 ampere-hours per ounce will give 4.2 ampere-hours per 100 square inches at a normal discharge rate. The ratio of weight of active material to active support should be as 1 to 1.9 and the active material should not be over 18 per cent of the weight of plate proper. The porosity, granular and molecular, should be such as to allow the mass of the active material to hold at least its own weight of electrolyte and thus allow for good diffusion of the acid. Active contact support should be at least 300 square inches per pound of plate proper and grid support 40 per cent of total weight of plate proper. The support contact should not be less than 1/3 that of the active material in thickness. The number of square inches of contact support per ampere of discharge should be no less than in the positive, and although the negative plate is, as a whole, of much lower resistance than the positive, a central diaphragm or wet conductor is of value as the principal starting point and distributer of the current. One all-important necessity for the negative grid is that it should be made in such a way that not more than 10 per cent of its superficial area be exposed to the electrolyte. Less than this amount is often advisable, since the larger the surface of the active material which is bathed by free acid, the better will be the maintenance of the negative voltage on high discharge work. The capacity of the positive should be 60 per cent that of the negative, so that the capacity output is limited by the positive, as working negatives down to low voltages is very deleterious.

Improvements of Accumulators.—The *Zeit. f. Elektrochemie* of April 15 contains a summary of suggestions concerning improvements in storage battery construction, from patent specifications issued in recent years. It is said that by far the greatest part of the patents issued in this line would not have been taken out if the inventors had had some experience in storage battery engineering. Only a small number of new patents refers to practically possible arrangements, and even these represent more or less suitable variations of well-known constructions. Large battery manufacturers do not care for such inventions, which are more or less beautiful in theoretical respect, but practically worthless; they prefer now as simple a construction as possible. Concerning the formation of the active mass, a few purely chemical processes have been suggested, while the majority refers to electrolytic methods. The latter, as far as they have proven effective, are based on the use of an anion, which is capable of forming a lead salt soluble in water. Under suitable conditions, lead peroxide is formed at once from lead, without a previous formation of lead sulphate. It is remarked that this interesting phenomenon appears to be in a near relation to the phenomena of the passive state of iron, chromium and nickel electrodes, and should be studied more carefully.

Influence of Light Upon the Formation of Accumulator Plates—In our March issue, page 117, we noticed some peculiar observations of Tommasi. In *L'Eclairage Electrique*, M. U. Schoop confirms Tommasi's results and describes the following experiment which he has made. He cut a lead peroxide plate, formed according to Planté, into three pieces, one of which was left in the dark, the second in the daylight, while the third was subjected directly to sunlight. He found that the color of the latter two changed decidedly after some hours. G. Rosset, in the *Centralblatt f. Accum.* sums Tommasi's results up in form of a single law, which is expressed as follows: Darkness favors the oxidation of lead, while light favors the reduction of lead oxide. This general law is a consequence of the phenomena of chemical equilibrium.

Storage Battery Traction.—H. Palitzsch describes in *Elektrische Bahnen*, April, No. 8, a new accumulator car used on the suburban lines of the tramway system of Dresden, Germany. The car, which has a capacity for 98 passengers, is provided with a battery of 184 double cells, each double cell having a capacity of 430 ampere-hours at a discharge rate of 140 amperes. The average discharge voltage is 365. The cells are charged from a station connected in series, the average voltage of charging being 480. When it becomes necessary to charge the cells from a 120 volt circuit they are connected in four parallel groups.

One of those German tramway systems which have used storage batteries on their cars for a long time, but have finally changed their system over to trolley, is that of Hanover. In the *Elektrotechnische Zeitschrift* of May 5, L. Liebenberg describes in detail the new trolly system and gives the following reasons for the change: great danger to the passengers from explosions of gases developed by the batteries; unreliability of the storage battery service; complications and expense involved in the attendance of the batteries.

Daniell Cell as Standard Cell for Technical Purposes.—In the *Centralblatt f. Accum.*, April 15th and May 1, G. Rosset remarks that the Daniell cell is still probably the most extensively used standard cell in technical laboratories. Since, however, the e. m. f. depends upon the concentrations, the e. m. f. is changed by diffusion, evaporation of the liquids, solution of zinc and deposition of copper, so that its e. m. f. is not constaht. For copper sulphate solution of specific gravity 1.2 or 1.1, zinc sulphate solution 1.2 or 1.4, Jenkin found the e. m. f. to be 1.099 and 1.069 international volts, respectively. Rosset has therefore endeavored to make the e. m. f. of the Daniell cell independent of concentration changes. He accomplishes this by use of equimolecular solutions of both sulphate and separating them from one another by means of a semi-permeable diaphragm of copper-ferrocyanide, which maintains the concentration of both solutions constant. He has found that under this condition the e. m. f. and the temerature coefficient are constant whatever the absolute concentration of the two solutions as long as they are equi-molecular. The e. m. f. is 1.105 as long as the solutions are not dilute, the temperature coefficient is —0.0002. He produces the semi-permeable diaphragm in an ordinary porous cup, the upper part of which he impregnates with paraffin; he then places the cup into water in order to remove the air from the pores, and fills it with a dilute solution of potassium ferrocyanide (about 4 per thousand); the cup is then placed in a dilute solution of copper sulphate (16 to a thousand). According to the greater or smaller porosity of the mass the diaphragm is then formed from 24 to 72 hours.

Miscellaneous.

The Continuous Steel Process.—The technical journals have lately contained a statement to the effect that the continuous method of making open-hearth steel, which is usually associated with the name of Mr. Benjamin Talbot, and the use of large tilting furnaces, had been adjudged by the United States courts to be the invention of Mr. Ambrose Monell. The statement is erroneous; the Monell process is for a *dis*continuous process in a fixed furnace, and the decision of the court simply affirmed Monell's ownership of his own invention, which Talbot had claimed to have invented also. The idea of the continuous process remains as Mr. Talbot's invention. Recently S. Surzycki has published accounts (*Stahl und Eisen*, Feb. 1,

1904; *Iron and Steel Magazine*, May, 1904), of having worked the Talbot process for two years, at Czenstocha, Hungary, in a fixed furnace, provided with two tap-holes at different levels, the lower one for draining the hearth when necessary, and normally kept closed, the upper for tapping out part, ordinarily half, of the finished steel in the furnace, as in the Talbot process. The rest of the process is as in the tilting furnace, ore is added to form an oxidizing slag, then pig iron, for a sharp reaction, then the bath brought again to tapping heat. The fixed furnace thus worked has its output increased some 15 to 25 per cent, with less wear on the hearth, and in brief unites the advantages of the Talbot continuous process with the cheapness and simplicity of the fixed furnace.

In *Stahl und Eisen* of April 15, 1904, O. Thiel refers to Surzycki's statements, and gives data concerning the application of the continuous process to fixed Bertrand-Thiel furnaces at Hösch. Two such furnaces of 27 tons capacity each, worked in this manner, gave 250 to 270 tons output per day, against 130 to 140 tons obtained from two 27-ton fixed furnaces worked continuously, but not on the Bertrand-Thiel plan, at Czenstochau. At Hösch, the upper furnace receives charges of 1 ton solid Thomas pig iron, 3.5 tons Swedish magnetic ore, containing 63 per cent iron and 1.1 tons of lime. The slag obtained contains 20 to 25 per cent of phosphoric acid. Tapping this metal into the lower furnace, it there receives 3.5 tons of scrap, 0.4 tons of roll scale and 0.7 tons of lime. The steel obtained is very pure, with 0.025 phosphorus, and averages 103 per cent in weight of the pig iron used. These latest developments in the continuous process promise to be of great importance in the open-hearth steel industry. The replacement of the second Bertrand-Thiel furnace by an electrically heated furnace, for the quick raising of the temperature to the tapping point, is well worthy the best attention of electrometallurgists.

Foundry Practice with Copper.—An illustrated article by W. J. REARDON, on practice with copper and its alloys in the foundry of the Westinghouse Electric & Manufacturing Co., is published in the March issue of the *Electric Club Journal*. For casting copper, the first requirement is that it must be heated to exactly the right temperaure, and that the sand must have the proper temper, neither too wet nor too dry. The hardest problem is to produce castings free from blowholes. For this purpose it is necessary to observe the following rules: Heat to exactly the right temperature; have the sand at the proper temper; melt the copper quickly; keep the metal covered at all times; never let the metal stand in the furnace when once ready; add a small amount of silicon-copper (see ELECTROCHEMICAL INDUSTRY, March, 1904, p. 121). In mixing and melting brass, the following simple rules should be observed: Weigh carefully; melt quickly and uniformly; prevent volatization as far as possible; pour at exactly the right temperature. At the present time the temperature is estimated by the naked eye in the Westinghouse foundry. A bearing metal high in lead resembling glyco-metal has been made in the Westinghouse foundry which gives excellent results. It is composed of copper, tin, lead and zinc, and is difficult to make, because of the tendency of the lead, whose specific gravity is high, to settle to the bottom; this difficulty was overcome by devising special methods of mechanical mixing. Another alloy was made up of the same elements, but the mixing was divided into two steps; the first being to make an alloy low in copper and the second to mix it with copper. The intermediate alloy was made by melting the copper first under a charcoal cover, and then adding lead, then zinc, then tin; overheating being carefully avoided. This intermediate alloy was poured into ingot moulds. After cooling it was put into melted copper. The resultant alloy could be cast in sand and withdrawn while still red-hot, without showing any signs of lead sweating out, although the temperature of the casting when withdrawn was far above the melting point of lead. The temperature must be watched very carefully, as a slight excess causes the lead to segregate. Sand for cores should be high in silica and low in alumina, bond being obtained with resin, flour and molasses. Sand for moulding should be fine and open. It should not contain much alumina, because the alumina (clay) bakes, preventing the rapid escape of gas, and so causes blow-holes and sponginess. An illustration is given showing a sand conveyor and moulding machine used in the brass foundry of the Westinghouse Electric Co.

Imperfect Equilibrium in Alloys.—F. Osmond contributes to the May number of *The Iron and Steel Magazine* (formerly the *Iron and Steel Metallurgist*) an account of some attempts made to ascertain the solubility of solid copper in solid silver. Alloys with 0.25, 0.50, 1, 2 and 4 per cent of copper were made, cooled rather rapidly, polished and examined under the microscope while being heated in air up to 500° C. Alloys with 0.25 and 0.50 per cent of copper do not differ from pure silver, showing the copper to be diffused in them in the state of a solid solution; but with one per cent of copper the saturation point of the solid solution is passed, for small particles of copper can be detected by their turning blue, while the metal around them becomes yellowish brown. This indicates a progressive concentration of the copper, an imperfect equilibrium. With more copper present, the eutectic appears, but the silver of the eutectic is shown to contain, by similar tests, an excess of diffused copper, besides its dissolved copper. There is thus shown in these cases a lack of uniformity in these solid solutions. The article concludes with the idea that when a substance in solution, with molecules of the gaseous type, is precipitated from solution at a temperature so low that diffusion is impossible, it may easily retain in solid substances the molecular or even the atomic type, since the isolated molecules or atoms lack the power to unite. The question is a wider one than that of the mere peculiarities of copper-silver alloys.

Manganese and Copper Alloys.—An article in the May issue of *Metal Industry* contains a summary of some investigations on the properties of alloys of manganese and copper.

Tungsten.—E. F. Smith and F. F. Exner presented before the American Philosophical Society the results of an investigation extended over a long period, on the atomic weight of tungsten. An abstract of the paper is given in *Science*, May 6. The authors doubt whether any chemists who in the past occupied themselves with the redetermination of the atomic weight of tungsten have worked with the pure substance. Tungstic acid is prone to form complexes. It was found that if the acid contained no iron, for instance, but be digested with acids, *i. e.*, hydrochloric or nitric acid, in which iron is present, the latter will enter the tungstic acid. Iron and manganese are eliminated from the acid with the greatest difficulty. In earlier work with tungsten, there is no evidence of the removal of iron and manganese nor of vanadium and phosphorus. The authors found that in purifying ammonium para tungstate by recrystallization alone, the tenth recrystallization showed vanadium. The slimy greenish or bluish-white masses believed to be "paratungstates" because of their great insolubility are probably complexes. The use of pure sodium carbonate (2 per cent) to dissolve tungsten dioxide gives an excellent means of ascertaining when the iron, manganese and silica are fully removed, but its development into a method for the determination of the atomic weight of tungsten is not at all probable. The plan of digesting pure ammonium paratungstate with nitric acid, then evaporating to complete dryness and gently igniting, affords pure oxide. Porcelain vessels are preferable to those of gold, silver or platinum for the ignition of ammonium paratungstate and tungstic acid. The oxidation of metal leads to reliable atomic numbers when the material is pure. Tungsten hexachloride can be completely transposed into pure oxide with water and a little nitric acid.

Radium from American Ores.—A paper on this subject was presented to he American Philosophical Society by A. H. Phillips, and an abstract of it is given in *Science*, May 6. The mineral from which the radium was separated was carno-

tite, a new mineral described in July, 1899, and found as yet only in the western part of Colorado and adjacent counties of Utah. In composition it is a potassium uranyl vanadate with three molecules of water of crystallization. The experiments of the author show that carnotite is a carrier of radium to a considerable extent, and would prove a valuable source of radium.

Filter-pressing in Western Australia.—The *Engineering and Mining Journal* of April 14 contains an interesting article by W. A. Pritchard, in which it is stated that more than 70,000 tons of ore and over 15,000 tons of accumulated tailings are treated monthly at Kalgoorlie by filter-pressing. This constitutes over 80 per cent of the output. Outside of Kalgoorlie, many of the mines of Western Australia are passing from 25 to 50 per cent of their ore through filter-presses. It will thus be seen that this process plays an important part in the extraction of gold. The filter-press in Western Australia has developed from a center-filling, hand-closing press, making one-inch cakes to a press filled on the side, closed by hydraulic pressure and making three-inch cakes. It is used solely for the purpose of extracting gold solutions from pulp, which is too fine for leaching, and for drying gold precipitates from zinc boxes. The gold solution, after filtering, is ordinarily pumped through clarifying presses to remove any fine slime which has passed through the cloths; it is then delivered to the precipitation boxes. In spite of unfavorable conditions for power, water and labor, the cost is not high. A pretty full description is given of the details of the method and of the construction of the presses.

De-tinning.—A note on a new process for winning tin from tin scrap is published in the April number of *Metal Industry*. Coke is crushed to a coarse powder and saturated with a strong solution of common salt, any excess of solution above that required to saturate the coke being drained off, so that the air is able to penetrate readily throughout the mass. The scrap to be stripped is covered with this moistened coke powder. The action commences very rapidly and is often complete in a few hours. To avoid patches of unstripped material on the surface of the scrap, it is well to cause the heap to be shaken from time to time, so that the whole of the tin surface of the scrap is brought into contact with the coke. When the black surface of the scrap shows that the separation is complete, the mass is washed with a jet of water, when it is found that the tin is washed out in the form of an oxide, probably in the form of hydrated stannic oxide, which forms a white milky precipitate in the water, from which it can be collected by allowing it to settle and subsequently filtering it off. No tin is found in the solution as chloride. The brine-moistened coke can be used over and over again, so long as the heap does not become too dry. The rough coke powder is separated from the stripped scrap by means of a screen, and when this is done the coke may be allowed to fall into unstripped scrap; the stripped scrap left on the screen being then well washed with a water jet, as a large amount of tin oxide adheres to the surface.

Determining Ozone in Gases.—A brief Faraday Society paper by E. G. P. Bousfield deals with the accurate determination of the percentage of ozone in gases not dissociated by moderate heat. It is published in the March issue of the *Electrochem. and Met.* The method is based on measurements of volume and the principle is to measure first a certain quantity of the gas containing the ozone; this gas is then passed over some substance which is capable of converting ozone into ordinary oxygen (like platinized asbestos, gently heated). On again measuring the gases, an increase of volume, due to the breaking down of O_3 molecules into O_2 molecules is found. From this increase of volume, the quantity and percentage of ozone originally present is readily calculated.

Copper Cyanide Solution.—The colorless solution of copper cyanide which is not precipitated by sulphuretted hydrogen, and which is familiar to all students of qualitative analysis, has been studied by J. B. Treadwell and C. von Girsewald, the results being given in *Zeit f. Anorganische Chemie*, and abstracted in *Eng. and Min. Jour.*, March 31. The formulas given in text books $K_2Cu(CN)_4$ and $KCu(CN)_2$ are apparently merely guesses. From a study of solutions containing various proportions of copper and potassium cyanide, the authors conclude that the salt present is $K_4Cu_2(CN)_6$, the ion not decomposed by hydrogen sulphide being determined as $Cu_2(CN)_6$.

Plating of Sheet Zinc.—After a review of the development of plating upon zinc, it is said in an article in the *Lead and Zinc News* of April 4, that in recent years the cost has been considerably reduced by plating the zinc sheets, with nickel before stamping and working them into shape. The sheets can be plated and polished at about one-half or one-twentieth the cost of the finished article. Zinc is the best base for the production of such sheets, as when cut it has a white edge which is very close to the color of the polished nickel. The cheapness of this base compared with brass and copper, to which it is equal in resisting oxidation, is a strong point in its favor. Sheet zinc coated with nickel is known in Germany as nickeline, but the industry has also been introduced into this country. The nickel is plated directly on the zinc without a previous coating of the zinc with copper. Zinc is also coated with copper and brass. The brass-coated article is often lacquered with a transparent lacquer, colored with a suitable coloring, and the finished article obtained with the color and appearance of gold. The copper finish is oxidized, spotted and lacquered, and then takes the place of the popular finish commonly known as antique or oxidized copper.

Firebrick.—A note in *Metal Industry*, April, refers to the selection of the proper kind of firebrick. It is pointed out that for a crucible furnace, hard firebrick is needed with a smooth surface and close grain, of the highest heat-resisting nature possible in connection with the other properties, and of a chemical nature, as near as possible to that of the clinker, so that there shall be no fluxing action, and consequently no abnormal adherence of the clinker to the brick.

Insulation for Wires.—In an article in *L'Electricien* of April 2, some uses of the Hackethal wire are mentioned. Its feature is the insulating material which is formed of a vegetable tissue impregnated with linseed oil and minium. The wire is stated to be used in telephone lines in Germany, and to have given good results in chemical works on account of its resistivity against the corrosive action of an atmosphere saturated with acids. If too strong a current is passed through the wire the insulating material dries out and is carbonized, but does not burn.

Fuel Economy from the Chemical Point of View.—A paper was read on this subject before the Liverpool Engineering Society by J. R. C. Kershaw, and a longer abstract is published in *Elec. Rev.*, of May 21. The author first deals with the fuel and water supply and recommends that, besides the steam-raising trials made with the fuel, thoroughly reliable samples of the fuel should be submitted to chemical and calorific examination, and that the results obtained in the trial under the boiler should be judged in the light of these laboratory results. He recommends an approximate analysis of the fuel, giving the per cents of volatile hydrocarbons, of ash and of fixed carbon. As long as the contract for the fuel supply runs, samples should be taken at intervals and submitted to analysis. The feed water should also be tested at quarterly or shorter intervals, since the amounts of lime and carbonate of soda or caustic soda for softening, depend upon the condition of the water. The author then deals with the combustion of the fuel in the boiler furnace. Perfect combustion of bituminous coal requires a sufficiency, but not an excess, of air; a sufficiently high temperature in the combustion chamber; and a perfect admixture of the air and volatile hydrocarbons. These requirements are discussed in detail. Examination of the waste gases is demanded; it should include the temperature of the exit gases, the draught behind

the chambers, and the chemical constitution of the gases. The author believes that if boiler engineers would pay more attention to the chemical side of their work, their annual fuel expenditure would be materially reduced.

Utilization of Peat and Lignite.—The question of the utilization of lignite and peat for power purposes is one of great commercial importance for various countries, and has been the subject of extended investigations. In an article in *Eng. and Min. Jour.*, April 7, H. H. WOTHERSPOON gives an account of recent advances made in this direction, especially in Germany. The principal deposits of lignite from which briquettes are made, exist in two different parts of Germany which include over 280 factories for the manufacture of the briquettes having a total of 680 presses. The output for 1902 is given as 12,438,000 metric tons; most of this fuel was consumed in the larger cities of Europe. The process of converting brown coal into briquette form consists in "developing the latent bituminous matter within the lignite by the aid of heat, and then compressing the same in powerful tube-presses which have a capacity of 60 tons per twenty-four hours." The wholesale selling price of these briquettes in the larger German cities is between $2.10 and $2.25 per metric ton. The brown coal of Alabama has been briquetted by this process quite successfully, and has been declared by the German experts to be superior to the best product from their own mines. The calorific value of these briquettes is between 7,500 and 9,000 B. T. U., depending upon the quality of the raw material. It is also stated that the advance in Germany in the manufacture of bog matter into fuel is quite remarkable. The rapid loss of water is stated to be due to "the mechanical processes through which the material is put on leaving the bog. The machinery which rapidly and thoroughly breaks up the cellular structure of the partly decomposed vegetable matter, sets free the water from the plant fiber and develops the peculiar vegetable mucilage or natural binding matter contained in the mass. After the peat has been thoroughly pulped, it is passed into compression tubes. Part of the moisture contents is here squeezed out of the mass, and the remainder evaporates rapidly on exposure to the air making a dense, clean, elastic fuel after about two weeks' time." Fuel bricks are clean to handle and practically smokeless, and they have a calorific value of between 8,500 and 10,000 B. T. U., Norton's figures, for the thermal value of American bogs range between 9,000 and 14,000 B. T. U. per pound, when bone-dry.

Castner.—In our January issue, 1903, we gave a reproduction of the Castner memorial tablet in Havermeyer Hall, Columbia University, and gave a brief account of the ceremonies of the unveiling of that tablet. The addresses delivered on that occasion are now given in full in the School of Mines Quarterly, January, 1904.

FARADAY SOCIFTY MEETING.

The May meeting of the Faraday Society was held on May 9 at the Institution of Electrical Engineers in London, Mr. Bertram Blount in the chair.

STUDIES IN VISCOSITY.

Dr. Charles E. Fawsitt delivered a paper on this subject, in the first part of which he discussed some relations of viscosity to salt formation. The viscosity of a mixture of two or more substances in aqueous solution is equal to the product of the viscosities which the solutions of the substances separately have. If, however, the substances act chemically on one another, then the viscosity of the mixture is different from the calculated value. By numerical illustrations from cases where a strong acid is neutralized by a strong base, this difference is shown to have a constant value for equimolecular quantities. This value corresponds to the formation of undissociated water from the hydrogen and hydroxyl-ions. The difference between the calculated and observed values in the case where a weak base is mixed with a strong acid has a value which varies with the base in the case of urea and its derivatives appears to vanish.

In the second part of his paper the author dealt with viscosity as an additive property. A comparison of the viscosity constants of some urea derivatives and aurides of the fatty series shows that for a difference in the composition of CH_2 there is a corresponding difference in the viscosity constant. There is, however, a pronounced "constitutional" effect as shown by the different values obtained for isomeric substances.

In the discussion which followed, Mr. B. Blount referred to the difficulties in making accurate and reliable viscosity measurements on which important deductions are based. Dr. T. M. Lowry discussed the relationship between viscosity and conductivity for varying temperature, and showed how, in the case of very dilute solutions, the conductivity-temperature curves converge to zero conductivity at 39° C., just as the viscosity curve for pure water does. Dr. Rudorf observed that with different forms of apparatus different results are obtained. Dr. Fawsitt in reply said that his results were all obtained with one apparatus and that they were strictly comparable. Referring to Dr. Lowry's remarks he added that viscosity is inversely proportional to migration velocity.

THE ELECTROLYTIC OXIDATION OF ANTHRACINE.

A paper on this subject by Mr. A. Fontana and Dr. F. M. Perkin was then read by Dr. Perkin. The authors have taken up the study of the oxidation of anthracine primarily to ascertain whether it is possible to obtain a good laboratory method for the preparation of anthraquinone. The first attempts were made with solutions in acetone, platinum electrodes being employed. Although oxidation took place in solutions of anthracine in acetone, it was not found possible to oxidize more than about 55 per cent of the anthracine. Attempts were then made to electrolyze anthracine suspended in 20 per cent sulphuric acid or in caustic alkali to which an oxygen carrier had been added. Various carriers were employed, the most satisfactory being chromium, cerium or manganese salts. The anode consisted of a lead vessel; the cathodes, four in number, were also of lead and were placed in four small porous cells. There was also a lead paddle which was connected with the positive pole, the total anode surface was about 6 square decimeters, the current density 1 ampere per square decimeter, the temperature 70° to 80° C., and the e. m. f. from 2.5 to 3.5 volts. The yield of anthraquinone was about 80 per cent, and it was not found possible to increase the yield more than 1 or 2 per cent above this. It was afterwards found that almost equally good results were obtained without a diaphragm. In this case the stirrer ½ square decimeter in area was made the cathode. In discussing the results of their experiments the authors consider that owing to its volatility and the comparatively poor yield a solution in acetone cannot be recommended. On a large scale no method in which a diaphragm is employed would be satisfactory, owing to the migration of the SO_4 ions from the cathode to the anode side. They then briefly review the patents of the Höchster Farben Fabrik, Darmstädter and Martin Moest. Le Blanc has criticised Darmstädter's patent, and doubted whether it would be possible to obtain any appreciable quantity of anthraquinone owing to the reduction of the chromate by the cathodic hydrogen. The authors are of the opinion that to a certain extent the chromium, managanese and cerium salts act as chromium does when employed in the electrolytic bleach industry, viz., by preventing reduction. Moest claims, in his patent, to obtain a hundred per cent yield of anthraquinone; the authors have been unable to obtain much more than 80 per cent; they are, however, of the opinion that very likely better results would be obtained on a manufacturing scale.

In the discussion which followed, Mr. Blount suggested the use of thallium salts. He thought the energy bill in this instance might not necessarily be an all-important item. Dr

Steinhart thought better results would be obtained if the anode solution were removed from the cathode. Mr. Inglis discussed the effect of the potential difference between the anode and solution. Mr. Gaster thought the cost of energy was of prime importance. He suggested the use of petroleum as a base for colors instead of tar.

Current Notes.

Philadelphia Section American Electrochemical Society. A meeting of the Philadelphia Section was held at the Engineers' Club on May 6. Dr. J. W. Richards read portions of his presidential address delivered at Washington, and led a discussion of some of the principle points, in which various members participated.

Titanium Trichloride.—With reference to recent French and English patents on the reduction of titanium tetrachloride to titanium trichloride, a correspondent calls our attention to a paper published by E. Polidoni in 1899 on the electric reduction of titanium tetrachloride to trichloride. (See *Zeitschrift f. anorganische Chemie*, v. 19, page 306; *Chemisches Centralblatt*, 1899, v. I., p. 518.) The industrial importance of titanium trichloride is due to its strong reducing properties, as well as to its use for etchings and dyeing leather, silk, etc.

American Electrochemical Society.—At a meeting of the Board of Directors, held on May 7, it was decided that a monthly bulletin be sent out by the Secretary, to include the names of applicants to be voted upon at the next Board meeting and those elected at the one preceding and as far as possible all notices that should be sent out for the current month. The first bulletin has just been issued. It states that the following gentlemen were elected members: Herbert Haas, Redding, Cal.; C. J. B. Henning, Santa Cruz, Cal.; Franz von Kügelegn, Holcombe Rock, Va.; F. H. Chapin, Minneapolis, Minn.; H. W. Buck, Niagara Falls, N. Y.; Henry N. Stokes, Washington, D. C.; Prof. F. H. Wolff, Jr., Washington, D. C.; H. W. Sykes, Syracuse, N. Y.; Charles Brandeis, Montreal, Canada; Charles H. Clarke, Berlin, N. J.; Horace H. Emrich, Perth Amboy, N. J. At the next meeting of the Board, action will be taken on the following applications for membership: George Breed, Philadelphia, Pa.; Alexander Lodyguine, Staten Island, N. Y.; Charles Jackson, Guadalajara, Mexico. The date of the St. Louis meeting is provisionally set for the 13th to 16th of September, inclusive. Sessions on two days will be held jointly with Section C of the Fifth International Electrical Congress, one sesion will probably be arranged to be held jointly with the Bunsen and Faraday Societies, and one independently.

Water-Power Developments.—A note in the twelfth annual report recently issued by the General Electric Co. should be of specially great interest to our readers. One of the important enterprises completed last year by this company is a power transmission plant at Guanajuato, Mexico. The power is transmitted at a continuous pressure of 60,000 volts. The transmission line is 101 miles long and the wires are carried on iron towers 48 feet high and 440 feet apart. Including steam and water-power plants, the General Electric Co. has installed and under contract at present 1,230,270 hp. capacity in polyphase generators. Of these, 514,910 hp. capacity are being operated by water-power.

Fixation of Atmospheric Nitrogen.—We understand that the Niagara Falls plant of the Atmospheric Products Co. has recently discontinued operation, but no authoritative information could be obtained whether this discontinuance of work is only temporary.

The Radioactive Atom.—Thomson, in a paper abstracted in our April issue on the structure of the atom, starts from the assumption that an atom is made up of electrons, and then goes on to determine by the rigid rules of mathematical physics, what structure must necessarily be attributed to the atom as consequence of this original assumption, and finally compares the results at least to some extent quantitatively with the experimental facts. In two recent pamphlets of A. Reuterdahl similar problems are discussed in a more speculative and qualitative way, the author giving a picture which he has made for himself on the structure of the atom to explain the facts, especially those on radioactivity and electrolysis. In a pamphlet on "the radioactive atom (Providence: *The Franklin Press Co.*, 1904) he gives, with the aid of diagrams, his views of the structure of the atom. He considers it as a highly complex structure, ultimately composed of very minute particles which he calls "energons" and assumes them to be simply "centers of activity." On account of the radioactive phenomena which he endeavors to explain, he calls his atom the "radioactive atom." It is made up of a spherical shell consisting of energons. Within the interior of the shell fairly stable groups of energons have room to exist. Groups of six energons he calls "dynamicons." When certain conditions exist, the energons constituting the shell have the power of attracting to themselves particular electrons from the space surrounding the shell; a rearrangement of the energons composing the shell takes place, and new energon groups may be formed, which pass into the interior of the shell. The author refers to Hovenden's theory of a substance or fluid which, when uninfluenced by external forces, rises from the surface of the earth or "anti-gravitates." But while Hovenden considered the rigid and incompressible ether to be this substance, the present author assumes it to consist of energon groups which possess a fair amount of stability, but may, nevertheless, be decomposed and undergo changes when subjected to the proper conditions. He applies the term dynamicon to this anti-gravitating substance. When a substance changes its state, for instance, from the liquid to the gaseous condition, the quantity of dynamicons within the radioactive atom also changes. Any increase in the number of dynamicons within the shell is accompanied by an increase of the number of energons constituting the shell. The expansion of the energonic shell begins, due to an excessive inflow of dynamicons, then the assimilative property of the shell also begins to exert itself and energons are added to it in such a compensating manner that the net attraction between the atom and the earth is at all times proportional to the established atomic weight. The chemical combination of atoms into molecules consists essentially in one atom entering into another, and becoming concentric with it. The radioactive atom may be considered as being provided with "polar openings." The alignment of these polar openings is termed "polarization;" it takes place when an e. m. f. is applied. The phenomenon of heat is a reaction between the energonic shell and the dynamicons. The phenomena of the electric current consist in a flow of dynamicons. The activity of the radium atom consists in continuously manufacturing dynamicons and other energon groups from surrounding energons and ejecting or radiating these manufactured products into the surrounding space.

Electrolysis According to the Energonic Hypothesis.—A. Reuterdahl's second pamphlet (Providence: *The Franklin Press Co.*) deals with this subject. He sketches the fundamental ideas of the dissociation theory and emphasizes that there must exist vast differences between the ionic and atomic conditions (a statement which has never been earnestly denied by anybody). When a solute is dissolved in a solvent, the author assumes the atoms and molecules to enter into reaction; there is a condition of a strain or tension between solvent and solute, which is due primarily to the ultra-atomic affinities between the energonic shells of the atoms and molecules involved ("herein lies the explantion of the real nature of osmotic pressure and solution tension; they are one and the same phenomenon, *i. e.*, ultra-atomic activity"); it is always the endeavor of every atom to absorb the maximum amount of dynamicons which its nature allows; equilibrium is reached when

the distribution is uniform throughout all the atoms and molecules involved. In certain well-defined cases there may occur an unlocking of the ions which constitute the molecule; this phenomenon "consists merely in a partial separation and increase in the distance between the adjacent energonic shells of the anion and cation. When this action is still further enhanced by the application of the electric current, for example, then the distance between the shells of the anion and cation may be augmented sufficiently to allow the cation to pass out of the anionic shell, thus becoming actually free and dissociated." The phenomena taking place in a simple concentration cell are explained by the author by the anology of the ordinary lift pump.

Correspondence.

ELECTROCHEMICAL NOTATION.

TO THE EDITOR OF ELECTROCHEMICAL INDUSTRY:

Sir.—In your issue of November last you published an editorial urging a thorough discussion of the advisability of a uniform international notation in electrochemistry in the columns of the engineering and scientific press, in advance of the international congresses to be held in St. Louis, this autumn.

I heartily concur with you in this matter. A fixed rule of notation should be established, but this cannot be done by simply adopting a resolution by a congress. Any recommendation should be thoroughly discussed in advance. There should be no ambiguity in future in the use of terms, like conductance and conductivity, or positive and negative plate of an accumulator; such handy abbreviations, like mol. for gram-molecule, should come into general use; and above all, when electrochemists decide to adopt a notation, they should adopt one which agrees with the notation of electrical and mechanical engineers. With respect to the latter point the table of the German Bunsen Society, on page 542 of your November issue, seems to be open to many objections.

What I wish to emphasize is that now is the time to start such a discussion and every electrochemical scientist and engineer interested in this subject should now express his opinion.

A. N. H.

New York.

[We fully agree with the sentiment expressed by our correspondent and will gladly open our correspondence columns to those interested in this important subject. ED.]

Book Reviews.

CHEMIKER KALENDER, 1904. By Dr. R. Biedermann, with an appendix. Berlin: J. Springer. Price, 4 marks.

This well-known German almanac for chemists, physicists, metallurgical engineers, etc., is now published for the twenty-fifth time. The general arrangement has not been changed since its last issue, while the results of recent researches have been made use of in correcting details. For the use of logarithms a short and clear explanation has been given; experience has shown that this is not superfluous for many chemists in laboratories as well as factories.

The tables of inorganic compounds (40 pages) and of organic compounds (124 pages) should be appreciated by American chemists and engineers, since they are arranged alphabetically according to the German name which is followed immediately by the chemical formula. These tables therefore form an excellent dictionary of chemical compounds which should be extremely useful for foreigners when reading German chemical literature. Besides the German name and the chemical formula, these tables give the molecular weight, notes on crystallization and color, the density (with respect to that of water or air as unity), the melting point, the boiling point and the solubility in cold or hot water, as well as in alcohol, acids and alkalies (for inorganic compounds) and the solubility in water, alcohol and ether (for organic compounds). Then follow extensive tables of the specific gravity of various solutions, of solubilities, and on analysis. It should be acknowledged that the advances made in electrochemistry have found due consideration.

DIE METALLE, Geschichte, Vorkommen and Kewinnung, nebst ausführlicher Productions-und Preis-Statistik: von Bernhard Neumann, Halle: W. Knapp. 421 pages. Price 16 marks.

The author is a professor of the Technical High School at Darmstadt, and the work was written to the order of the German Society for Trade Improvement, and afterwards awarded a prize by them.

The work is the outcome of industrious compilation and abstraction from larger works, and as such has a certain value. It deals with all the metals, even the rarest ones, and therefore will not entirely disappoint the reader in any line. The *History* is in each case necessarily brief, but nevertheless well written up to about 1900; from there on, unreliable. The antiquarian part of the history has received most attention. The *Occurrence* in nature is very briefly dismissed. The *Manufacture* is, like the history, more complete in its antiquarian aspects, and less satisfactory in regard to the processes now actually in use. The *Statistics* of price and production form the part of the book really of the most value, for the compiler has collected from many difficultly-accessible sources data which are seldom given in books on metallurgy, such as price and production in ancient and medieval times, and so on continuously to the year 1902. Numerous clearly-drawn and well-lithographed plates enhance the value of this part of the work.

To the electrometallurgist or chemist, the book will be valuable, principally for the statistics; although some interesting data may be, perhaps, extracted from the antiquarian part of the history and processes of manufacture, the rest of the work will not have any claims on his attention. There is a class of readers, however, to whom the work as a whole, will be particularly valuable, viz., to students of political economy and industrial economic development. For such, the treatment is well-balanced and the data accumulated of great importance, and in a form easy to understand and utilize.

SODA CAUSTICA, cloro e clorati alcalini per el trolise: per Ingegnere Fabio Villani. Milan: Ulrico Hoepli (Hoepli Manuals). 16 mo., 314 pages. Price 3.50 lire.

This is a beautifully printed book, as are all the Hoepli manuals, and contains much useful information. The work is divided primarily into two parts, electrolysis by the wet way and the dry way (fused salts), but as the former covers 300 pages, and the latter only 15, we begin to suspect a lack of balance in the treatment. This impression is deepened when we search the book for the detailed description of processes, and find *only* that of the *"Villani* electrolytic bath, with continuously depolarized anode," the process using managanese dioxide as an anodic depolarizer. As far as the reader of the book would be instructed by it, the processes of Castner, Kellner, Hargreaves and Bird, Acker, etc., in fact, all except that of Villani, might as well be non-existent.

If the book be taken simply for what it is, viz., a description of Villani's apparatus, and a detailed account of the information which he made use of and that which he acquired in conducting his experiments, then the reader need not be disappointed. The chapter on the volumetric analysis of the solutions used, and especially that on the control and chemical surveillance of an electrolytic plant producing caustic soda and bleach, are replete with valuable practical information for anyone working in this branch of manufacture. The specialist in this business who can read Italian will find the book certainly well worth getting.

Die Elektrolytische Raffination des Kupfers. By Titus Ulke. Translated into German by Viktor Engelhardt, Halle: W. Knapp, 1904, 152 pages, illustrated.

This is the tenth volume of the Monographs for Applied Electrochemistry, issued by this well-known German publishing house. It is a translation of Ulke's well-known book on copper refining into German. The original American edition was reviewed in our number of December, 1903, page 594. Nothing appears to have been added in the German translation, which evidently has been well and carefully prepared.

TWO STORIES OF ZINC.

A very neat pamphlet issued by the Lanyon Zinc Company contains two well-written stories of zinc which give a very clear exposition of the general principles governing the various steps in the reduction of spelter from ores and in the making of sheet zinc.

The "Story of Spelter" is first told. The principal ore supplies of the Kansas zinc smelters come from the Joplin district. On arrival at the smelter they are transferred to huge bins, which hold hundreds of tons. There the ore is dried. Afterwards it is crushed to uniform size to be treated in the roasting or calcining kilns. Calcining or desulphurizing is a simple but very important process during which the zinc is changed into zinc oxide, while the sulphur passes off as sulphur dioxide. The collection of the fumes and their subsequent condensation into sulphuric acid affords a handsome source of income to the United Zinc & Chemical Co., the only smelter west of the Mississippi River which makes any attempt to save this product. Several devices are described for stirring the ore, so as to expose each little particle to the oxidizing action of the heated air. It is not necessary to maintain an extremely high heat in the calcining kilns, a temperature of 1700° F. being the general average in use. This temperature is not kept uniform throughout the length of the kiln, but is gradually increased as the period of calcination progresses.

From the calciners the ore is conveyed to store houses where it is graded according to metallic contents. As needed for smelting, the ore is conveyed to a "mix-room," where it is mechanically mixed with finely ground coal and fairly coarse coke or coal, as reducing agent. The mixture is then treated in the distillation furnace which is described at some length and illustrated, while a sketch is given of a day's work in a distillation furnace plant. Despite many efforts of the metallurgists of the world, the losses of zinc in the process of distillation are greater than the losses of extraction of any other metal. The losses average from 10 to 30 per cent of the metallic contents of the ore and are largely due to the passage of zinc into the materials which compose the retorts, the escape of zinc fumes through the pores of the retorts and cracks therein or by the breaking of the retorts, producing entire loss of the charge in these receptacles. Some of the zinc vapors escape through the condensers and still another loss is incurred in cleaning out the retorts, while preparing them for a new charge, the latter loss being unavoidable.

The "Story of Sheet Zinc" is next told. Some historical data on this industry in the United States are first given. It is mentioned that the operation is one requiring skilled help and constant watching, lest any one of a score of contingencies arise. Only the choicest of spelter can be used in sheet zinc manufacture. Once the selected slabs of spelter reach the zinc rolling mill they are fed into a huge melting pot which has a capacity of 15,000 to 35,000 pounds of molten semi-fluid metal. From this huge pot the metal is withdrawn by the aid of huge ladles and poured into moulds, usually 10 by 30 inches, and an ingot or billet is cast, having a thickness from one to three inches, depending upon the size and thickness of the sheet to be rolled. After the zinc has had time to cool in the moulds the billets are passed on to a room in which they are annealed at a temperature of about 300° F. A moderate change in the temperature of the annealing oven will cause a variation in the character of the sheet produced from the annealed metal. This peculiarity is a subject little understood by the average consumer of sheet zinc, and were he to take advantage of this fact, he could frequently secure sheets more perfectly adapted to his particular business by advising the rolling mill of the particular use to which he intends putting the sheets; they would then be made accordingly.

From the annealing oven the billets go direct to the first or "roughing" rolls. The sheet is then sent to power shears—great knives which clip off the edges of the sheets of metal. They are then cut in half and are passed through a second annealing oven where they are subjected to a temperature of from 350° to 400° F. for a period of about two hours, and then gathered into "packs" consisting of 18 or 20 sheets, and in these packs sent to the finishing rolls. The finest and most delicate step of the process is the finishing. The margins of the sheets are finally again trimmed off by power shears and the sheets are inspected for flaws, and are packed.

The use of sheet zinc in the United States has been growing steadily for years. From a total production of about 3000 tons of sheet zinc in 1873, the consumption has increased more than ten-fold. Among the more common uses of sheet zinc are the manufacture of metallic ceilings, stove boards, the linings of refrigerators, fruit jars, tops, etc., and large quantities are used in the precipitation of gold from cyanide solutions. The use of small galvanic batteries takes quite a large tonnage of this metal in the course of a year, since it has been demonstrated that batteries and rods made from sheet zinc are of a more satisfactory character than those which are cast. To-day for 90 per cent of the newspaper illustrations of the United States, or half-tones which are made, sheet zinc is used. Paper and fine cardboard makers, as well as lithographers, use large sheets in their various lines of work.

Since sheet zinc is capable of very excellent treatment through the process of turning and stretching, these sheets are to-day entering into formidable competition with sheet brass and copper, being plated with copper or brass as the articles are made up. The manufacture of nickel, copper and brass plated zinc sheets is as yet comparatively young in the United States, although the prospects for this line of work are very excellent. Skilled workmen are constantly developing new uses for sheet zinc, and there is to-day no reason to doubt that the next decade will see the demands for sheet zinc more than doubled.

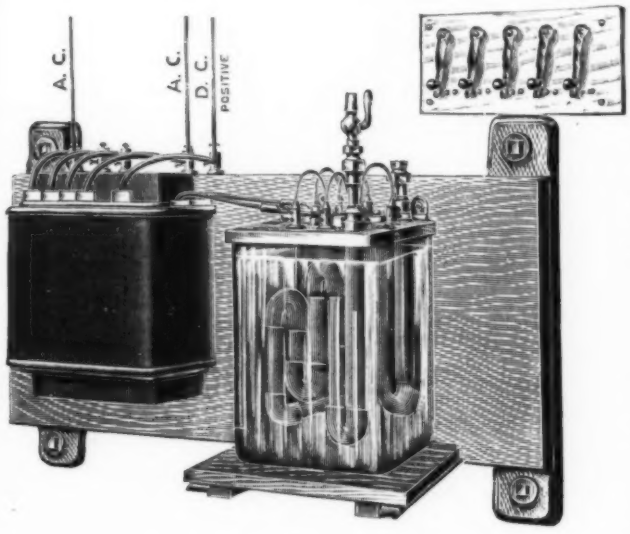
The pamphlet is well printed and nicely illustrated with various pictures in the text, while two larger illustrations are added, one showing the Iola plant of the Lanyon Zinc Company, which is the largest zinc smelting plant in the United States, and the other giving a typical view of a zinc furnace in operation, showing the tongues of zinc and other vapors escaping from mouths of the condensers. A map of the United States zinc mining camps is also given and the towns are marked in which zinc smelter are located.

ELECTROLYTIC RECTIFIER.

In our February issue (page 76) we published a description of the Churcher electrolytic rectifier, made by the Electric Appliance Co., of Cincinnati, Ohio. Its characteristic feature is that it is used in connection with a transformer, the secondary of which is tapped in the center, and is there connected to a platinum plate in the rectifier, while the two terminals of the secondary are connected to two aluminium plates. During one half-wave of the alternating current, one-half of the transformer is active; during the next half-wave the other half-wave, and so on.

The adjoining illustration shows an electrolytic interrupter, also made by the Electric Appliance Co., of Cincinnati. This interrupter gives promise of extensive application in the X-ray field, as it is a well-known fact that alternating current is very inefficient for such purposes, especially when the frequency

is over 60 cycles. The Churcher alternating rectifying interrupter will receive any potential, any phase or cycle of alternating current and deliver a rapidly and completely interrupted current through the primary of an induction coil. Its construction and mode of operation is similar to the

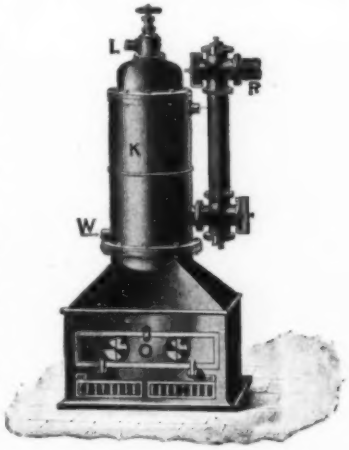


ELECTROLYTIC RECTIFIER.

Churcher rectifier, the only difference being in the area of exposed contact of the positive electrode, which, in this case, is of restricted area; restricted to such a degree that instantaneous interruption of the current is obtained by the well-known Wehnelt effect. Adjustment is possible by the regulation of the amount of positive electrode exposed.

SULPHUR FURNACE.

The adjoining illustration represents a sulphur furnace, made by the Schutte & Koerting Co., of Philadelphia, the applications of which are quite manifold. It is especially applicable for bleaching sugar, starch, glue, gelatine, etc., in the chemical industry, further for sulphurizing and preserving hops in breweries; and for disinfecting purposes.



SULPHUR FURNACE.

This furnace consists of four main parts. First, the bottom part, the pan in which the sulphur burns. A regulating device is here provided for controlling the supply of air for burning the sulphur.

The second part is the cooler K, which is cast in one piece, and through which water flows to cool down the gases, so that the latter do not reach the saturating tanks at too high a temperature.

The third main part is the cap covering the cooler and the fourth main part is the exhauster working by steam or compressed air, according to the circumstances, to draw the gases through the cooler and to discharge them under pressure through the outlet pipe, or suck them through the outlet pipe.

According to the application, either one or the other method is used. In some cases it may be advisable to have the steam mix with the sulphur gases, and then the exhauster is placed behind the tanks to be sulphurized. The furnace and cooler are made of cast iron, and by taking off the top cap the whole apparatus, especially the pipes in the cooler, through which the gas flows, is exposed for cleaning and renewals. The exhauster is fitted with a spindle, both for adjusting the discharge pressure and (within certain limits) the supply of air to the furnace.

The advantages which the Schutte & Koerting Co., as manufacturers, claim for this sulphur furnace, are low price, simple construction and small space required; certainty and regularity of working, absence of variation in volumes of gas produced; absence of moving parts, with a resulting saving in wear and tear; no necessity for skilled attendance.

INDUSTRIAL NOTES

THE FOOTE MINERAL CO., of Philadelphia, has recently issued a nicely illustrated new abridged edition of their catalogue of mineral collections, individual specimens and laboratory minerals, for schools, colleges, teachers, chemists, assayers, students, mining experts, etc.

We have received from THE LEEDS & NORTHRUP CO., of Philadelphia, their illustrated catalogue of electric testing instruments, dealing with condensers, galvanometers, bridges, resistance boxes, testing sets, potentiometers, etc. The description of the Leeds & Northrup potentiometer should be of special interest to electrochemists.

THE SCIENTIFIC MATERIALS COMPANY, of Pittsburg, Pa., has just issued the 1904 edition of its voluminous catalogue of apparatus and supplies for chemical and biological laboratories. The catalogue comprises 436 pages, and is one of the most complete lists of chemical laboratory apparatus which exists. The arrangement is alphabetical, according to the names of the apparatus, so that it is very easy to find what one is looking for. Each apparatus is briefly characterized and its main features are pointed out. Very often the apparatus are illustrated. The list of balances alone comprise 37 pages.

PERSONAL.

MR. WOOLSEY MCA. JOHNSON, of the Lanyon Zinc Co., was recently in New York.

MR. C. O. BAKER, president of Baker & Co., incorporated, of Newark, N. J., and New York City, the well-known refiners and artisans in platinum, gold and silver, sailed on Wednesday, May 18th, on the White Star Steamship "Cedric," for his annual trip abroad.

The St. Louis World's Fair has started and foreign engineers and scientists are beginning to arrive. Among the gentlemen whom we had the pleasure to see in our office were Dr. O. Weil, chief chemist of the thermit department of the Goldschmidt works in Essen; Mr. F. Froelich, who represents the Verein Deutscher Ingenieure at the St. Louis Exposition, and Dr. Leopold Rostosky.

DIGEST OF U. S. PATENTS

PRIOR TO JULY, 1902.

Compiled by Byrnes & Townsend,
Patent Lawyers,
National Union Building, Washington, D. C.

PRODUCTION OF ALUMINIUM FROM FUSED ELECTROLYTES.

(*Concluded from page 210.*)

527,846, October 23, 1894, Leonard Waldo, Bridgeport, and Frank A. Gooch, New Haven, Conn.

Injects hydrogen into fused baths of aluminium compounds which liberate chlorin and fluorin when electrolyzed. The

hydrogen combines with the halogens and the resulting hydrochloric and hydrofluoric acid gases are removed by a draft flue over the vessel, or by endless belts of fabric running through water and over the vessel. The hydrogen also agitates the bath, protects its surface from the action of the atmosphere and prevents corrosion of the anodes by the halogens. The hydrogen may be introduced through a tubular carbon anode. A platinum anode may be used. The preferred bath comprises: Cryolite, 55 parts; aluminium chloride, 12 parts; magnesium chloride, 5 parts; sodium chloride, 28 parts. The bath is replenished with aluminium chloride only. E. M. F., 4 to 10 volts.

527,847, October 23, 1894, Frank A. Gooch, New Haven, and Leonard Waldo, Bridgeport, Conn.

Introduces water into fused baths of aluminium compounds which liberate chlorin or fluorin when electrolyzed, to remove the halogens as acid gases. Appartus same as in 527,846. Steam may be injected near the anode or through a tubular anode; or a water spray or small pieces of ice may be dropped under the bath. Steam tends to oxidize the carbon anodes. The preferred baths are: (1) Aluminium sodium chloride, 15 parts; cryolite, 55 parts; sodium chloride, 30 parts; or (2) Cryolite, 55 parts; aluminium chloride, 12 parts; sodium chloride, 19 parts; potassium chloride, 14 parts. Bath replenished by aluminium chloride only. E. M. F., 4 to 10 volts.

527,848, October 23, 1894, F. A. Gooch, New Haven, and Leonard Waldo, Bridgeport, Conn.

Removes chlorin and fluorin by adding water, as in 527,847, employing the same apparatus. May substitute water-gas for steam. The preferred bath is: Cryolite, 55 parts; aluminium chloride, 12 parts; magnesium chloride, 5 parts; sodium chloride, 28 parts. Replenishes with aluminium chloride only. E. M. F., 4 to 10 volts.

527,849, October 23, 1894, F. A. Gooch, New Haven, and Leonard Waldo, Bridgeport, Conn.

Similar to the preceding patents, but removes the chlorin or fluorin as acid gases by adding, from time to time, hydrated oxalic acid or other compounds containing either water of crystallization or of constitution or both. Baths: (1) Aluminium sodium chloride, 15 parts; cryolite, 55 parts; sodium chloride, 30 parts; or (2) Cryolite, 55 parts; aluminium chloride, 12 parts; sodium chloride, 19 parts; potassium chloride, 14 parts. E. M. F., 4 to 10 volts. The oxalic acid tends to oxidize carbon anodes.

527,850, October 23, 1894, F. A. Gooch, New Haven, and Leonard Waldo, Bridgeport, Conn.

Same as 527,849, except the bath, which is: Cryolite, 55 parts; aluminium chloride, 12 parts; magnesium chloride, 5 parts; sodium chloride, 28 parts. Bath replenished with aluminium chloride only. E. M. F., 4 to 10 volts.

527,851, October 23, 1894, F. A. Gooch, New Haven, and Leonard Waldo, Bridgeport, Conn.

Like the preceding, but removes the chlorin or fluorin and replenishes the bath by adding, from time to time, hydrous aluminium chloride. Bath: (1) Aluminium sodium chloride, 15 parts; cryolite, 55 parts; sodium chloride, 30 parts; or (2) Cryolite, 55 parts; aluminium chloride, 12 parts; sodium chloride, 19 parts; potassium chloride, 14 parts. E. M. F., 4 to 10 volts. Carbon anodes are somewhat oxidized.

528,365, October 30, 1894, F. A. Gooch, New Haven, and Leonard Waldo, Bridgeport, Conn.

Same as 527,846, except baths, which are: (1) Aluminium sodium chloride, 15 parts; cryolite, 55 parts; sodium chloride, 30 parts; or (2) Cryolite, 55 parts; aluminium chloride, 12 parts; sodium chloride, 19 parts; potassium chloride, 14 parts.

562,785, June 23, 1896, Heinrich F. D. Schwahn, Kansas City, Mo.

Mixes aluminious minerals purified by the process of his patent 514,039, with nitric, hydrochloric and sulphuric acids, and alkaline earth salts and oxides, *e. g.*, earths and barium sulphate, magnesium or calcium carbonate or dolomite, calcium hydrate or calcium or magnesium chloride. Adds sodium chloride or fluoride or cryolite, and sawdust or carbon. Heats to a glow, powders, mixes with carbon, preferably charcoal, places in an iron crucible and passes electricity through the mass. The mixture may be roasted in a combustion furnace before effecting reduction by the electric current. The apparatus employed is a horizontal, revoluble muffle of iron, surrounded by a heating jacket which is surrounded by an air jacket. The iron muffle serves as one electrode, two carbon rods of opposite polarity to the vessel passing through its side. The current employed is from 3500 to 5000 amperes, at 35 to 50 volts.

578,633, March 9, 1897, Frank A. Gooch, New Haven, Conn.

Electrolyzes a fused bath of: (1) Sodium chloride, 100 parts; hydrated aluminium chloride, 125 parts; and alumina, 8 parts; or (2) Sodium fluoride, 10 parts and hydrated aluminium chloride, 200 parts. The excess of aluminium chloride is decomposed by the heat of the bath into alumina, water and hydrochloric acid. The bath is replenished by alumina. E. M. F., 4 to 10 volts.

605,380, June 7, 1898, Henry S. Blackmore, Mount Vernon, N. Y.

Melts equal parts of sodium and potassium sulphides; adds sodium and potassium thiocarbonates; and finally adds alumina, which reacts to give aluminium sulphide which melts and becomes part of the bath. Electrolyzes with carbon anodes, the aluminium depositing at the cathode and the sulphur combining with the carbon anode to give carbon disulphide, which combines with the alkali sulphides present to reproduce the thiocarbonates. May produce the thiocarbonates for the initial bath by introducing carbon disulphide into the bath or producing it therein by the action of introduced sulphur on the carbon anodes. The reactions, involving the production of carbonyl sulphide, are illustrated by equations.

631,253. August 15, 1899. F. A. Gooch, New Haven, Conn.

Melts together an aluminium compound and a haloid of a metal more positive toward sulphur than aluminium. Adds a sulphur compound and alumina and electrolyzes. Specifically: Dissolves alumina in a fused bath of sodium fluoride, three parts, and aluminium chloride, two parts; covers the bath with powdered carbon; injects carbon disulphide or hydrogen sulphide, converting the alumina into aluminium sulphide, and electrolyzes with a current of sufficient voltage to overcome the resistance, plus 0.9 volts. May produce carbon disulphide in a bath covered with charcoal by introducing sulphur. May use an initial bath of aluminium fluoride and sodium fluoride. The apparatus comprises an iron cathode vessel lined on the bottom and half-way up the sides with carbon, the remainder of the vessel and anode openings in the sides being lined with "compacted" alumina. Carbon tubes passing through the side openings serve as anodes and inlets for the carbon disulphide. An iron bell lined with carbon depends into the bath to carry aff the gases.

673,364. April 30, 1901, William Hoopes, New Kensington, Pa.

Purifies aluminium from silicon, iron, copper, magnesium, calcium, sodium and potassium, by passing a current from an anode of the molten impure metal through a fused bath of cryolite, the silicon, iron and copper remaining undissolved on account of their lower affinity for fluorine than aluminium, while the other impurities dissolve with the aluminium, but are not redeposited with it, on account of their lower affinity for fluorine. The bath may contain alumina and may consist of chlorides of aluminium and any metal more positive than aluminium. The apparatus comprises a carbon-lined cathode vessel, in which is suspended a carbon anode tray containing the impure, molten aluminium, and having a terminal covered by a carbon tube. A current having a density of 20 to 30 amperes per square inch of the aluminium anode surface, is employed.